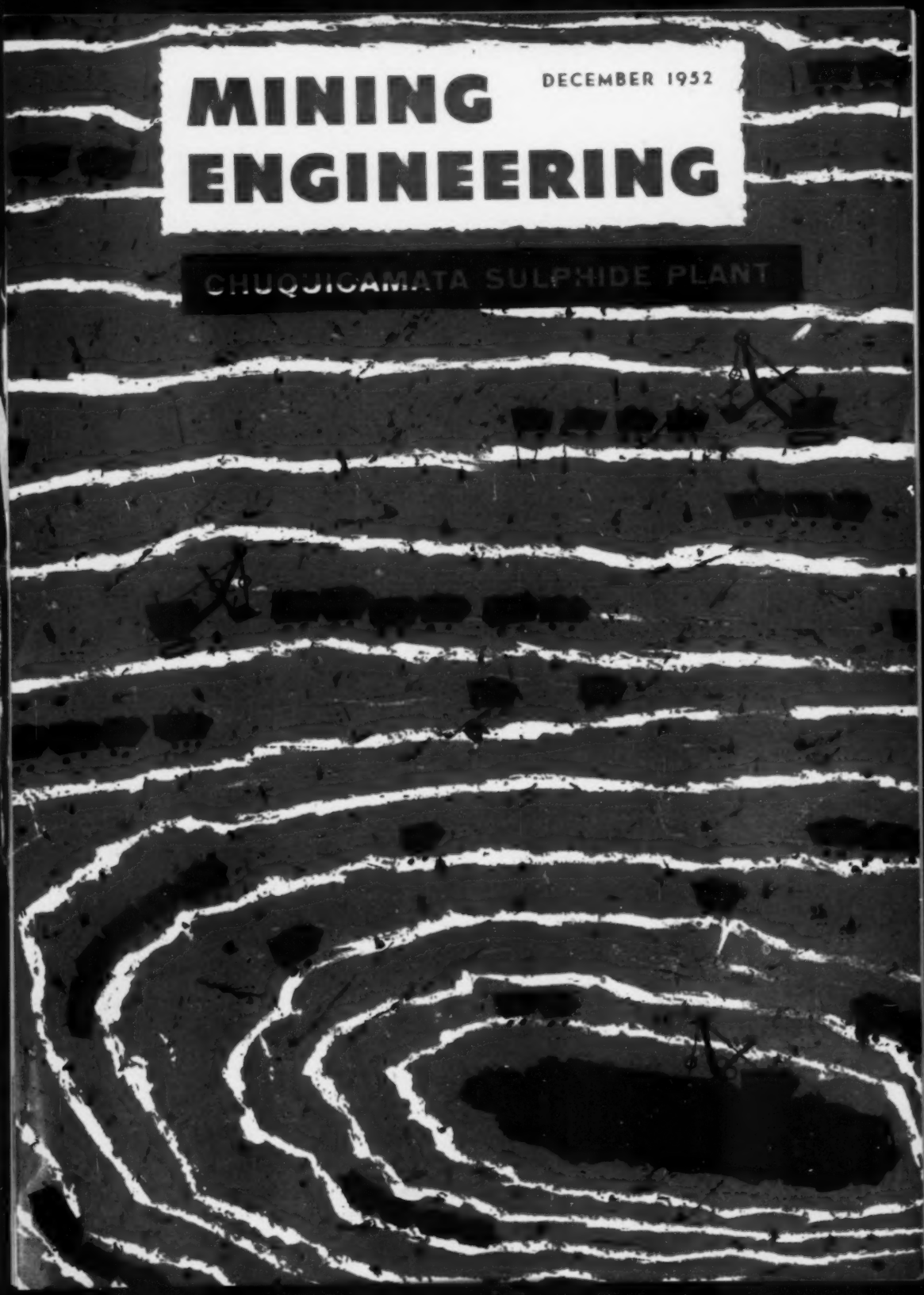
MINING ENGINEERING

DECEMBER 1952

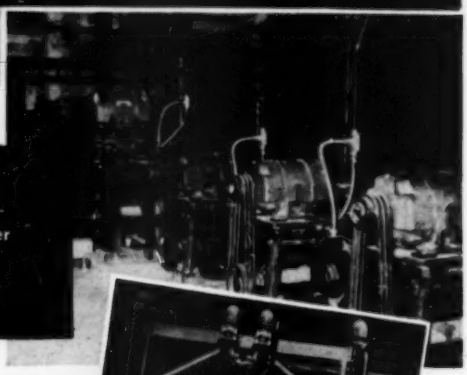
CHUQUICAMATA SULPHIDE PLANT



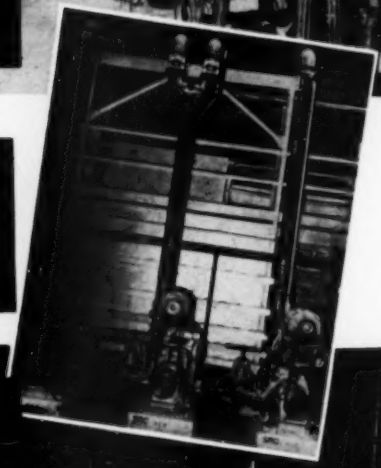
WILFLEY *Sand* PUMPS

In the World's Greatest Open Pit Copper Mine and Sulphide Plant at Chuquicamata, Chile

Wilfley Sand Pumps **are an integral part of the huge operation carried on in the fabulous new $100,000,000 plant at Chuquicamata, Chile. Day and night dependable Wilfley's deliver continuous, trouble-free performance . . . playing a major role in the dramatic success of this great mining enterprise.**



Primary Rougher Overflow Pumps—Secondary Rougher Overflow Pumps in background.



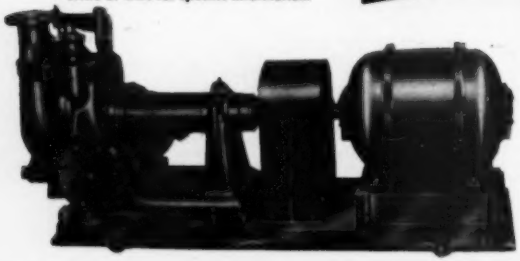
Secondary Cleaner Tails Pumps—Secondary Cleaner Overflow Pumps in background.

Final Concentrate Pump—Secondary Rougher Overflow Pumps in background.

New Model "K" WILFLEY *Sand* PUMP

The new Model "K" Wilfley Sand Pump answers the 3 important economic considerations in the handling of solids: **1.** High Operating Efficiency—the Model "K" features important mechanical improvements which give you substantial dollar savings in power and production costs. **2.** Low Maintenance Cost—Wilfley's precision engineering assures dependable operation with a minimum replacement of parts. **3.** Minimum Installation Expense—the Model "K" is specially designed for simple installation, saving plant operator's time, trouble and added expense.

Write or wire for specific information.



A. R. WILFLEY & SONS, Inc.

Denver, Colorado, U.S.A.

New York Office: 1775 Broadway, New York City, U.S.A.

MINING ENGINEERING

Incorporating Mining and Metallurgy, Mining Technology and Coal Technology

VOL. 4 NO. 12 DECEMBER, 1952

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Chuquicamata Sulphide Plant

COVER

Chile Exploration Co.'s great open pit is symbolized in copper by cover artist Herb McClure. Engineers realize that men, machines, and money must be skillfully interposed between ore-in-place and refined red metal. The story of Chuquicamata as it begins to mine its great store of sulphide ore is told starting on page 1163.

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— Personnel Service —

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Engineer, 43, married. Sixteen years' experience mining. Desires permanent position mine superintendent. Broad knowledge all types underground mining and shaft sinking. Construction and milling experience. Prefer North American continent but will accept foreign. Presently located Australia but available May 1, 1953. M-706.

POSITIONS OPEN

Mining Engineer with broad exploration experience, young, single status, and must be in excellent physical condition, as nature of work will be under very primitive conditions. Salary about 25 pct more than applicants last salary rate plus travel expenses and living allowance. Duration until July 1953. Location, Burma. Y7498.

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Three Mining Engineers wanted for exploration and open-pit type iron ore mining in Malaya.

Box J-20
MINING ENGINEERING

ECONOMIC GEOLOGIST

For geological field investigations. Must be capable of independent appraisals of mineral prospects. Prefer MS or equivalent. Require minimum of three years' field geological experience involving evaluation of mineral deposits and some experience in petrographic work. Midwestern location as operational headquarters. Progressive industrial organization, employee benefits. Submit complete resume including photograph and salary requirements.

Box J-21
MINING ENGINEERING

Mining Engineer, graduate, with experience in underground operations for a fluorspar mine. Salary open depending on experience. Location, western United States. Y7724.

Mining Engineer for limestone mine, young, graduate, with several years' practical experience, preferably in the operation of a draft operation mine. Must have supervisory ability. Company engaged in a modernization program involving the construction of a new three thousand barrel per day cement plant. Will put into practice the recommended methods for new and more efficient mining techniques, and will supervise future operations. Location, Missouri. Y7609.

Engineers. (a) Chief Engineer, graduate, mining, who has had 5 to 10 years' experience in surface and underground surveying, drafting and ore reserve calculation planning of surface and underground installations. Two year contract. Location, Central America. (b) Jr. Geologist, who has had some field experience, either during summer vacations or 1 year after graduation, for large mining company. Salary, $3600 a year to start. Location, Central America. Y7589.

Director of Engineering for an engineering institute, 30 to 40, with Doctorate in mining, chemical, electrical or mechanical engineering. Must have good experience in research, preferably industrial, and good teaching record. Salary, $6000 to $7000 a year. Location, South. Y7166.

"GIGANTIC WAR SURPLUS EQUIPMENT SALE"

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William Campbell Fellowships

Applications are invited for the Krumb Scholarships and Campbell Fellowships for the academic year 1953-1954. These are available to candidates for degrees in the fields of Mining, Mineral, and Metallurgical Engineering.

The Krumb Scholarships, with yearly stipends of $1000 and transportation to New York City, are in support of studies toward the B.S., M.S., or professional engineering degrees. Preference is given to students in mining engineering in the ratio of three in this field to two in metallurgical and mineral engineering.

The Campbell Fellowships, established to encourage research in the general field of metals, are available to candidates for graduate and professional degrees including the Ph.D. Five grants are usually awarded each year, with stipends of $1800 for single students and $2100 for married students.

Application blanks and further information may be obtained on request to the Office of University Admissions, Columbia University, New York 27, N.Y. Complete applications should be returned before February 20, 1953.

Project Mining Engineer with at least ten years' experience in design, layout and cost estimating for new mines and production plants. Should be capable of supervising construction on all such projects. Must have rather broad practical and technical knowledge of mining and milling operations. Salary, $6000 to $6600 a year. Occasional travel. Location, Texas. Y6976.

Engineers. (a) Mill Superintendent with minimum of five to ten years' experience in mill for a metal mining company. (b) Assistant Mining Engineers, for metal mining operations in the Philippine Islands. Salaries open. Y5022.

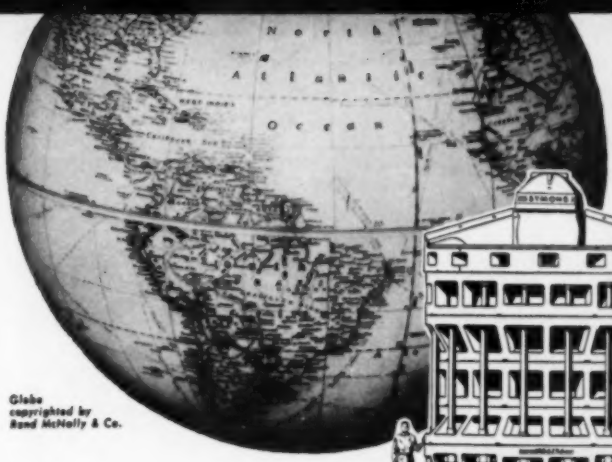
General Superintendent for tin and tungsten operation in Alaska. Must have excellent operating background. Will take complete charge of mining, milling and exploration programs. Must have good production record. Labor force is Eskimo. Salary, about $10,000 a year with house furnished free. Board will be $2.50 a day. Will be given a participating contract up to 2 pct of the net before taxes and depletion. Y7766.

Exploration Geologist, under 40, with at least five years' prospecting, examination, mapping and drilling experience for metal mining fields. Must speak Spanish. Salary, $5000 to $7000 a year. Location, Cuba. Y7510.

POSITIONS AVAILABLE

The Bureau of Mines is recruiting mining engineers, metallurgists, and fuel technologists, to serve abroad on mining projects conducted under the Point Four and other foreign programs. Salaries range from $5,000 to $9,000 per annum, depending on the qualifications of the applicant, the responsibilities of the assignment, and the area in which the work is done. There are also rental, and in some areas other allowances to compensate for unusual conditions encountered abroad. Some short-term assignments are available from time to time but most of the positions carry a minimum two-year tenure of office. Travel expenses for the employee are paid under all circumstances and for the longer-term assignments travelling expenses for family and transportation of a limited quantity of household goods are provided. United States citizenship and security clearance are mandatory for applicants, and United States citizenship for wives who accompany their husbands abroad also is required. Applicants must be physically capable of performing rugged work in the field. Additional information can be obtained upon application to the office of the Regional Director, Region IX (Foreign Minerals), Bureau of Mines, Washington 25, D. C.

WHEREVER MINERALS ARE PROCESSED...



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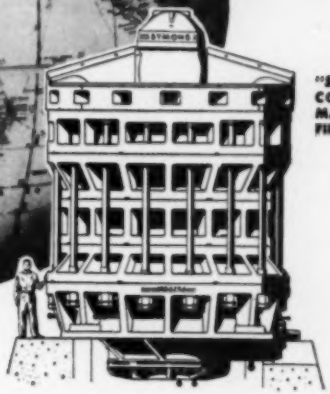
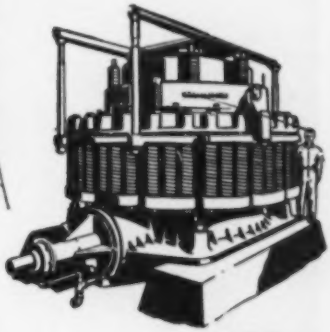
for BASE and PRECIOUS METALS ...the RARER ELEMENTS... METALLIC and NON-METALLIC MINERALS



"SYMONS" GYRATORY and CONE-CRUSHERS for PRIMARY, SECONDARY and FINE REDUCTION CRUSHING



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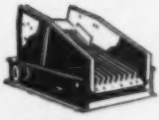
is the leading choice of producers THE WORLD OVER...



"SYMONS" VIBRATING GRIZZLIES and SCREENS for SCALPING and SIZING

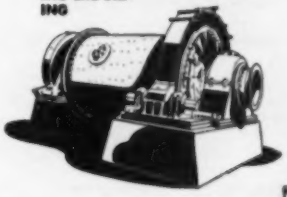
"SYMONS" "V" (VERTICAL) SCREENS for EXTREMELY FINE SEPARATION

- Without the vast mineral processing operations in all parts of the globe, all industrial capacity would be seriously hampered. And without efficient, large-capacity machinery, ore and mineral processing operations would fall far short of their required output.

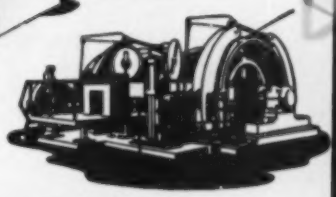
It is highly significant that wherever mineral resources are found in quantity—efficiency minded producers are now using, or are in the process of installing, Nordberg Machinery.

This dependable Nordberg Machinery includes Mine Hoists; *"SYMONS"* Gyratory Crushers for primary breaking; *"SYMONS"* Standard and Short Head Crushers for fine reduction crushing; *"SYMONS"* Vibrating Grizzlies and Screens for scalping and sizing; Grinding Mills for wet or dry grinding and a complete line of heavy duty Nordberg Diesel Engines in sizes from 10 to over 10,000 H.P. in a single unit.

Write for literature on the machinery you need.



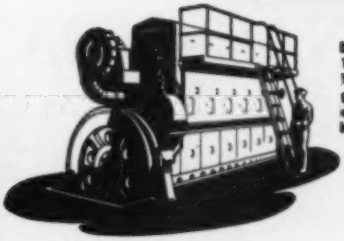
BALL, TUBE and ROD MILLS for WET and DRY GRINDING



MINE HOISTS

NORDBERG MFG. CO., Milwaukee, Wis.

"SYMONS"... A NORDBERG TRADEMARK KNOWN THROUGHOUT THE WORLD



DIESEL ENGINES, 10 to over 10,000 H.P. Burn gas, oil, or any combination of both.

C752-R2

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25,000 TON SULPHIDE PLANT
CHUQUICAMATA, CHILE

BUILT BY

FOLEY BROTHERS, INC.

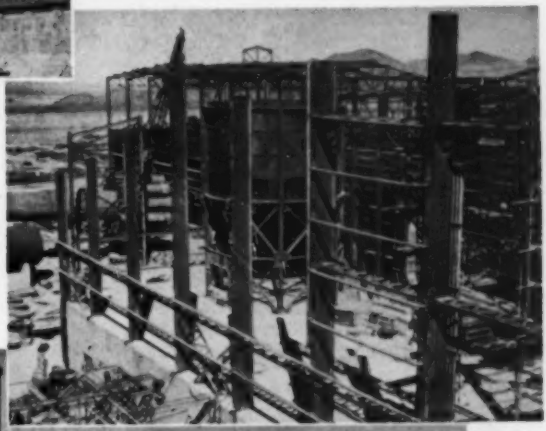
FOR THE

ANACONDA COPPER MINING COMPANY

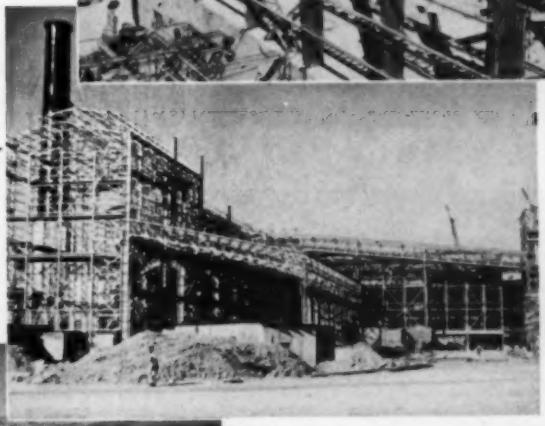
FOLEY BROTHERS, INC.

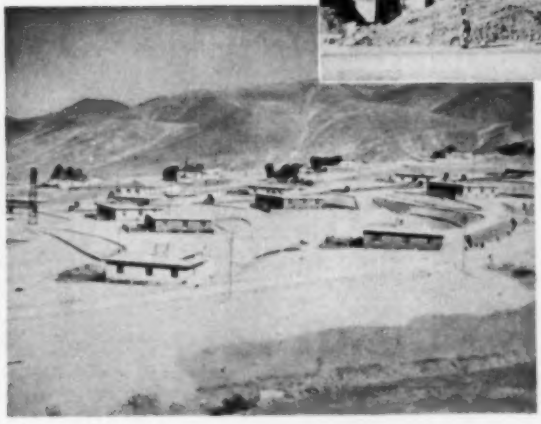
PLEASANTVILLE, NEW YORK

INTERNATIONAL CONSTRUCTORS



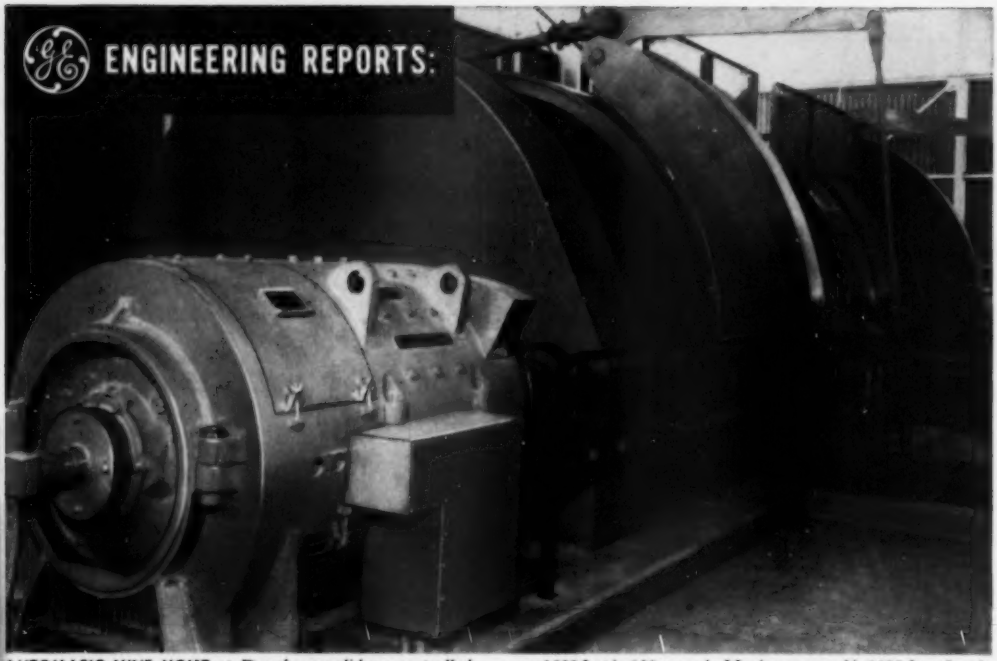
OVER 75 YEARS IN HEAVY CONSTRUCTION





Views of construction activities during the development of the sulphide plant, Chuquicamata, Chile.

ENGINEERING REPORTS:

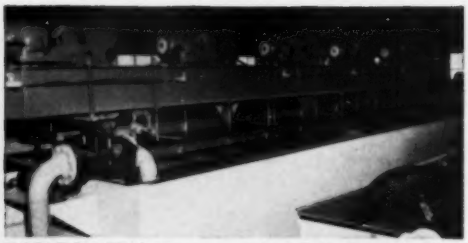


AUTOMATIC MINE HOIST at Duval—amplidyne-controlled and driven by a 600-hp G-E MDP motor—hoists 6 tons of ore up 1628 feet in 103 seconds. Maximum speed is 1100 fpm. Loading, acceleration, deceleration and dumping are all automatic.

Co-ordinated electric equipment

ROD MILL inside refinery is driven at 19 rpm by a 720-rpm G-E Tri-Clad* synchronous motor rated 250 hp, 440 volts.

*Registered Trade-mark of General Electric Company.

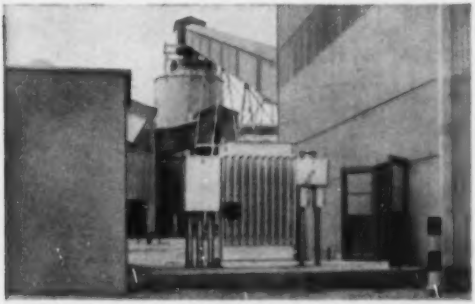


FLOTATION CELLS shown are agitated by five 5-hp G-E motors, totally enclosed, fan-cooled against corrosive vapors.

CENTRIFUGES, another step in the process, are driven by these 150-hp G-E Tri-Clad wound-rotor motors.

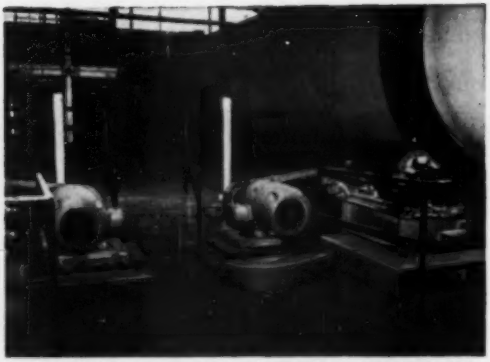
CONTROL OF MAIN ORE HOIST can also be manual by means of master control located in this hoist console.

POWER purchased by Duval is stepped down from 67,000 to 4160 volts at this G-E 5000-kva master unit substation.



DEPENDABLE G-E TRANSFORMERS step down 4160 volts to 480/277 volts grounded-neutral at 6 utilization areas.

keeps Duval potash output high



DRYING KILNS, where the refined potash is prepared for shipment, are driven by 40-hp G-E totally enclosed motors.

System engineered for efficient operation of automatic hoist and refinery drives

Typical of the way General Electric helps mining companies handle high tonnages at low cost is the new potash mine and refinery of the Duval Sulphur and Potash Company at Carlsbad, N. M. General Electric supplied power transformation equipment, refinery motors, and mine-hoist electric equipment.

G-E engineers—working closely with engineers of Duval Sulphur and Potash Co. and their engineering contractors, Stearns-Roger Mfg. Co.—applied this co-ordinated drive equipment to meet Duval's needs most efficiently and economically.

To put this kind of engineering leadership to work on *your* ore mining or processing problems, contact your local G-E Apparatus Sales Office—early in your planning. General Electric Co., Schenectady 5, N. Y.

660-27

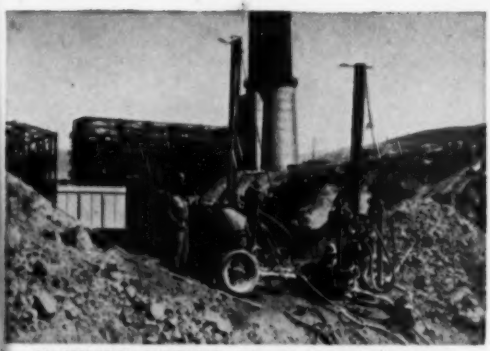
Engineered Electrical Systems for Ore Processing

GENERAL ELECTRIC

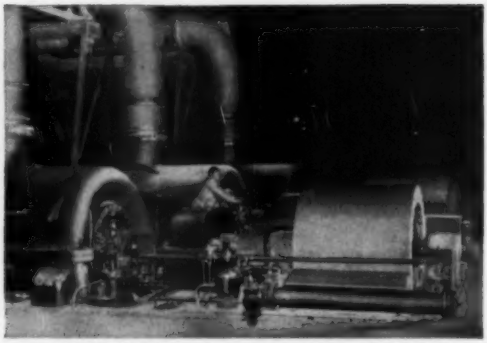
another milestone of

New sulphide to boost

In the foothills of the Andes, about 150 miles northeast of Antofagasta, at Chuquicamata, lies the world's largest known copper ore body. Owned and operated by the Chile Exploration Company, this great open-pit mine since its start of operation in 1915 has produced over 10 billion pounds of copper.

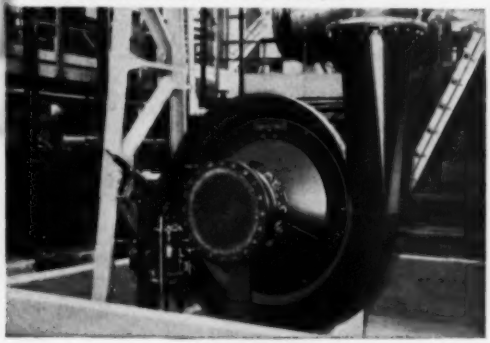


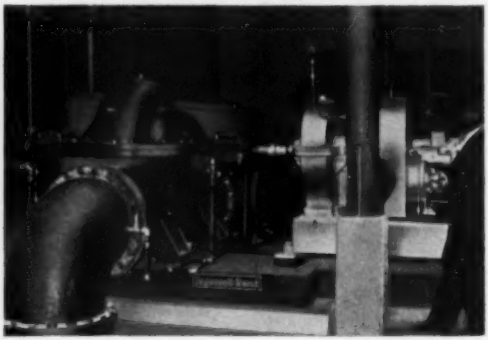
CONSTRUCTION EQUIPMENT – Where there's construction, there's rock to be removed. And at "Chuqui," Ingersoll-Rand Wagon drills and Jackhamers played a major role in the excavating. I-R portable compressors furnished air power for the drills.



CONVERTER BLOWERS – Three of the four Ingersoll-Rand 30,000 cfm, 15 lb. copper converter multi-stage Turbo Blowers which will serve the smelter at the new Chuquicamata Sulphide Plant.

FLOTATION BLOWERS – One of the five Ingersoll-Rand Type FS Turbo blowers that will serve the flotation plant.





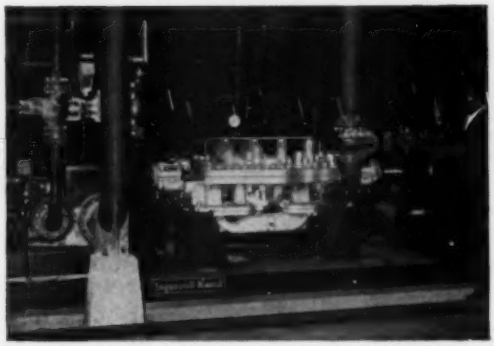
CIRCULATION PUMPS – A 20-inch Ingersoll-Rand Class LFV Condenser circulating pump. Three additional identical units are also installed. Each pump is rated at 10850 gpm.

COMPRESSORS **IR** AIR TOOLS • ROCK DRILLS • TURBO BLOWERS • PUMPS • CONDENSERS

progress at Chuquicamata...

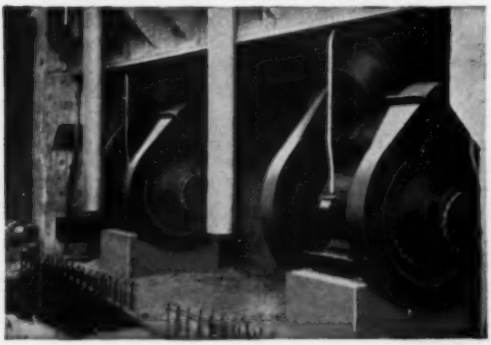
ore-processing plant output of strategic copper

Ingersoll-Rand equipment used both in the construction and the operation of Chile Exploration Company's new plant.



BOILER-FEED PUMPS – Feed water for the boilers at this new installation will be handled by four I-R Class RT 8-stage pumps. Each pump is rated at 167 gpm and 575 psi.

REVERBERATORY BLOWERS – Eight FS Turbo blowers like these pictured here will serve the Reverberatory furnace oil burner. The casting furnace will use four additional FS blowers.



Strategic copper will get a big boost in production in early 1953 when the Chile Exploration Company's new copper sulphide plant at Chuquicamata goes into full operation. This new modern plant was designed to increase the present Chuquicamata output to more than 500,000,000 pounds of copper per year.

New crushing plants, a concentrator and a smelter are among the principal structures being erected, not to mention a large-scale housing development and a new sewage disposal plant.

Ingersoll-Rand compressors, wagon drills, Jackhamers and other tools have been used extensively in the construction of these new facilities. And I-R blowers, compressors and pumps will be used in their operation.

But the name Ingersoll-Rand is not new in the Chuquicamata district. I-R equipment has been on the job at "Chuqui" since the start of operation there in 1915.

And in all parts of the world, wherever materials must be mined, you'll find Ingersoll-Rand.

• DIESEL ENGINES

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854—14

ENGINEERING DATA

EQUIPMENT: 2 Hewitt-Robins Reclaimers (200 TPH Cap. ea.) serving 3 storage beds 50' x 360' (11,000 tons Cap. ea.).

LOCATION: Chuquicamata Smelter of the Chile Exploration Company at Chuquicamata, Chile, S.A.

MATERIAL HANDLED: Copper concentrates of varying analysis (35% avg. cu. content), reverts and lime rock flux.

PERFORMANCE: Charge of uniform analysis is fed to 2 furnaces on a continuous basis to provide copper matte of maximum quality.

HEWITT-ROBINS BLENDING SYSTEM ASSURES UNIFORM CHARGE FOR CHUQUICAMATA SMELTER

Continuous smelter output at top capacity and maximum quality control of product are assured when a furnace charge is prepared with the Hewitt-Robins Bed Blending System. For this system permits furnace operators to maintain a uniformly predictable charge at all times . . . even though the analysis of the charge ingredients may vary widely.

Constant uniformity is obtained by bedding all the raw materials—both copper bearing elements and fluxing agents in predetermined proportions—in hundreds of superimposed horizontal layers. Then, by reclaiming the materials cross-sectionally, they are thoroughly blended and ready for furnace feed.

This means that charges can be prepared on a large scale . . . loss of copper in slag is minimized . . . frequent sampling and changes to the makeup of the charge are unnecessary.

Results prove that the Hewitt-Robins Bed Blending System provides the best method for obtaining chemical and physical uniformity in raw materials which are variable in analysis.

For complete information write for Bulletin No. 152

HEWITT-ROBINS (HR) **INCORPORATED**

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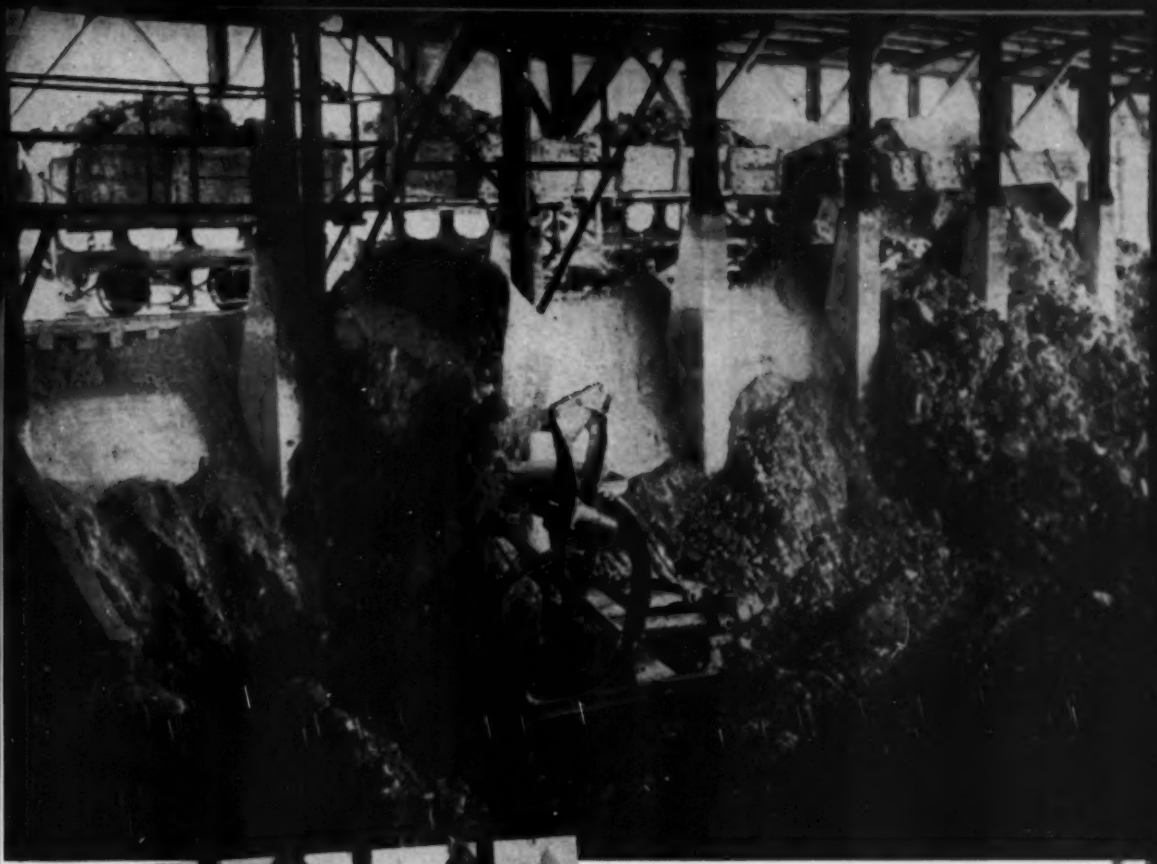

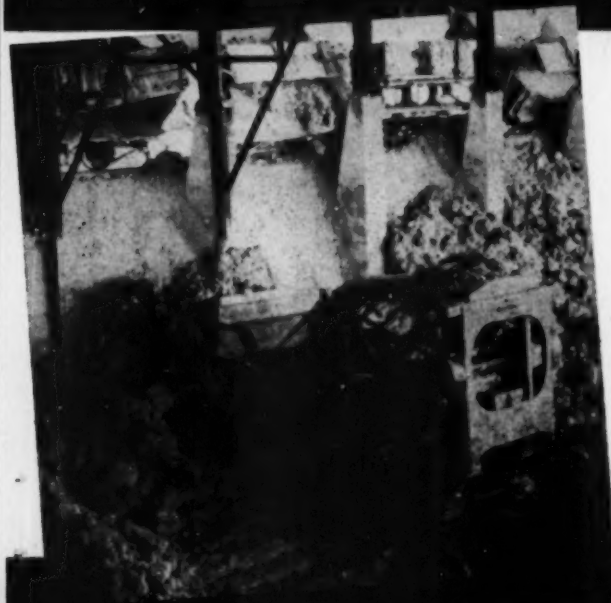

STOCK PILE LOADING

Loading clay with an Eimco 104 at a Southern clay operation has averaged 800 tons per shift for the better part of a year.

The clay material discharges easily and leaves the bucket clean.

This typical Eimco 104 loading job is repeated on many other types of material; rock, sand and gravel.

For economical, fast loading investigate Eimco loaders first.

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IN ENGLAND: EIMCO (GREAT BRITAIN) LTD., LEEDS 12, ENGLAND

IN ITALY: EIMCO ITALIA S.P.A., MILAN, ITALY

You Can't Beat an Eimco

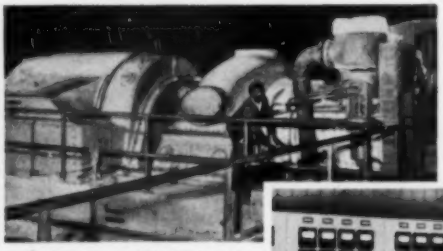
Again, Westinghouse

WESTINGHOUSE is proud to have supplied a substantial part of the electrical equipment for the vast new modernization project that will boost copper output at Chile's Chuquicamata Mine to over 250,000 tons a year. This equipment joins other Westinghouse apparatus already hard at work for the Anaconda Mining Company in the job of supplying copper-hungry defense and civilian industry in the Americas.

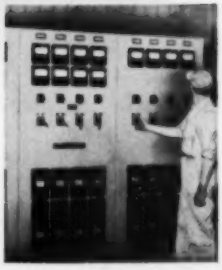
Wherever you look at Chuquicamata you'll see the Circle W insignia—notably in the huge new sulphide ore smelting plant, heart of the modernization project. Westinghouse equipment on the job here includes all gear motors . . . most induction and wound rotor motors . . . high and low tension substations and other key apparatus. In addition, three Westinghouse 7500-kw waste heat turbine-generators furnish about 50 percent of the plant's power. Also lending a hand is a fleet of Westinghouse combination trolley-diesel electric locomotives that haul ore from working face to smelter.

Westinghouse earned this assignment at Chuquicamata because of its reputation for efficiency and dependability . . . because its equipment is paying off every day for mine operators in many parts of the world under the toughest operating conditions. No matter how big the job is, or what the special problems involved, Westinghouse experience can help show you the way to *more profitable* mining operations.

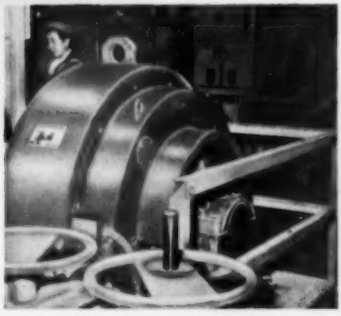
● 85-ton Westinghouse Whitcomb double-duty locomotive at Chuquicamata. Operates on overhead, third rail, or diesel engine.



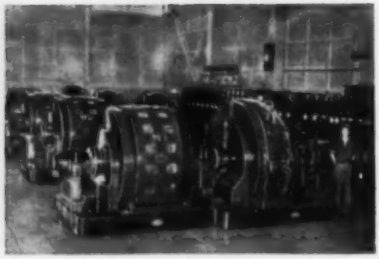
● Close-up of Westinghouse 7500-kw turbine-generator in smelter power plant.



● Westinghouse Control Board for concentrators.

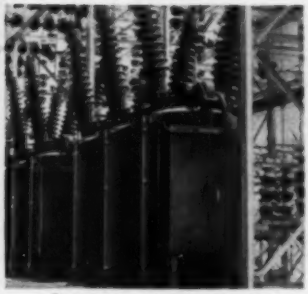


● 300-hp Westinghouse synchronous motor driving regrind mill in concentrator.

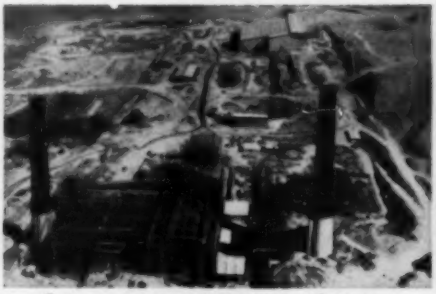


● Westinghouse M.G. sets in substation No. 5, oxide plant. This equipment, installed in 1926, is still going strong.

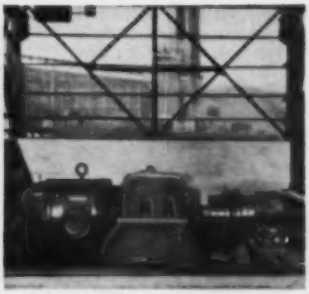
helps shoulder the load!



● Circuit breakers at concentrator sub-station No. 10. Type G.M.-2,138,000 volts, 800 amps.



● Air view of new $111,000,000 sulphide plant.



● Westinghouse gearmotor driving conveyor in smelter.

POWER SUPPLY—Westinghouse makes a complete line of substation equipment; from transformers and breakers for large outdoor units to packaged power centers for installations near the load center. Also steam and hydro turbine-generators . . . all types of transmission equipment . . . and engine-driven generators.

MINE HOISTS—Motors, controls, switchgear and transformers are manufactured by Westinghouse for balanced and unbalanced shaft or slope hoisting from any level.

Westinghouse supplies a complete line of equipment for every mining operation

Westinghouse designs and manufactures electrical equipment to speed movement of ore through every stage from working face to refinery. Flow chart of typical mining operations shows Westinghouse equipment you can use to lower costs and increase production.

MINING—A complete line of motors, generators, and controls for shovels, dredges and draglines; for slushers, crushers and other mine equipment. Clamps, hangers, splicers, etc., for trolley line installations and maintenance.

CRUSHING—GRINDING—Mine operators use Westinghouse equipment for grinding and crushing operations, including wound-rotor motors for jaw crushers, across-the-line and reduced voltage starters, high voltage starters; speed reducers, synchronous motors for rod and ball mills and ball mill feeder regulators.

CONCENTRATION—Famous for dependable service are Westinghouse Type CSP Life-line motors. Vertical and horizontal gear-motors for driving flotation cells and all types of conveyors in each concentrating process. Factory-assembled control centers are provided for economical, centralized operation.

SMELTING AND REFINING—Westinghouse makes transformers for all electric furnace duty. Also Rototrol for automatic regulation from indications of arc current and voltage; MC mill motors; large MG and Rectifier sets for ac-dc power conversion in tank house operations—and equipment for all electric smelting.

FOR MORE INFORMATION about these products check with: ELECTRIC UTILITY AND TRANSPORTATION DEPARTMENT, WESTINGHOUSE ELECTRIC INTERNATIONAL COMPANY 40 WALL STREET, NEW YORK 5, U. S. A.

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MEET THE AUTHORS

FRED D. DeVANEY

Fred D. DeVaney (*Magnetic Roasting of Iron Ores*, P. 1219) has presented several other AIME papers, all of them concerned with grinding and concentrating problems. He is chief metallurgist, Pickands Mather & Co., at Hibbing Minn., after working for the U. S. Bureau of Mines at Tuscaloosa, Ala., and Rolla, Mo., as a junior metallurgist and a metallurgist. DeVaney started as a research metallurgist with Pickands Mather in 1942. He set up the first laboratory in the Lake Superior district devoted to taconite concentration, his major interest. He has been an AIME member since 1923, holding various offices, including the first presidency of the Minnesota Section. DeVaney has a Bachelor's in Mining, and Master of Science earned at the University of Alabama in 1924. He is a member of the Engineers Club of Northern Minnesota and plays golf for relaxation and enjoyment.

MARSTON G. FLEMING

Marston G. Fleming (*Effects of Alkalinity on the Flotation of Lead Minerals*, P. 1231) is deeply interested in mineral activities in North America. He won the Nufield Traveling Fellowship in 1951, spending four months visiting universities, research laboratories, and plants in the U. S. and Canada. He has authored several other papers, including *The Paymaster Retreatment Plant, the Trend of Mineral Dressing in North America, and the Effects of Soluble Sulphide in the Flotation of Secondary Lead Minerals*. He is presently senior lecturer in mineral dressing at the Royal School of Mines. He joined the school after four years as a navigator with the Royal Canadian Air Force during World War II. Born in Montreal, he attended Queens College, receiving a B.S. in Mining and Metallurgy. Later, Fleming earned a Ph.D. in London. He has held underground and mill positions with various companies.

Jack Swart, member of the Explorer's Club, mein host, photographer, chief engineer for Foley Brothers, construction contractors for the sulphide plant at Chuquicamata, and man about mining, knows the world as only a man who spent most of his life seeing it, can. At Chuqui, he is on call day and night, but somehow, he manages to take visitors to Tocapilla where his boat is moored, for some fishing. He also managed to take practically all the photographs which appear in this issue. He filled the requirements of the authors, advertisers, and editorial department.

His membership in the Explorer's Club is genuine because he's spent more time out of the U. S. than in it. Born in Denver, Colo., Nov. 25, 1899, he started roaming the map early; Africa, England, Peru, and Chile are some of the places he has

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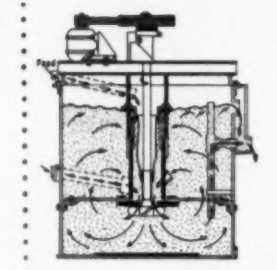
More Effective Conditioning IS possible with DENVER Patented SUPER-AGITATOR and CONDITIONER

Denver SRL Pumps

Note how recirculation ports in standpipe eliminate short-circuiting. Adjustable collar on standpipe gives better recirculation control.

Shutdown is no problem. Rubber-covered wearing plate prevents sanding up.

Two of the reasons why so many mills use DENVER Super-Agitator and Conditioners are: (1) more substantial construction (2) greater flexibility in operation.

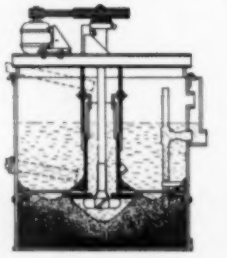


Denver-Buckman Concentrators

SIZES:
3' x 3' to 20' x 20'— also pilot plant and batch laboratory sizes. Standard and acid proof construction.

Remember, quantity production means low cost and good deliveries.

Write today.



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JOHN A. SWART

consin, attended Cornell, Navy Engineering School and the University of Minnesota. He holds a Master of Science in Metallurgy. Stowasser is currently in charge of the Allis Chalmers taconite pilot plant. His career started with the A. O. Smith Corp., as a metallurgical researcher. After two years with Allis Chalmers in the process machinery laboratory, he served as a lieutenant, J. G. with the Navy in destroyer engineering. He returned to Allis Chalmers following service. This is his first published paper.

R. J. Piros (*Theory, Scale-up, and Operating Variables of the Peterson Top Feed Reservoir*, P. 1236) has won a national championship for bait casting and several titles in his home state, Missouri. He presently lives in a fisherman's paradise, Lakeland, Fla. Piros received a Bachelor of Science in Chemical Engineering from Northwestern University. A member of the varsity football team at Northwestern, he won his letter in 1945. A junior member of AIME, he is now with International Minerals Chemicals' Research Div.

reached in his wanderings. And he's worked on some of the more famous mining enterprises in the U. S., too. He started while attending the University of Minnesota, working on the first taconite project during summers at Babbitt.

CLAIBORNE VAN ZANDT

Claiborne Van Zandt (*Features Incorporated in the Design of Lone Star's Two New Dry Process Cement Plants*, P. 1244) graduated from the University of Colorado with a B.S. in Engineering, after election to Alpha Tau Omega, social society, and Eta Kappa Nu, honorary electrical fraternity. Now a resident of Scarsdale, N. Y., he can be found on Long Island Sound sailing before the breeze, when business commitments permit. He is a member of AIME with an experience resume that begins with employment as a laborer on a machinery erection gang for the Colorado Portland Cement Co., at Ft. Collins, Colo. The following summer, 1927, he moved up to draftsman for the same company. Today, he is vice president in charge of engineering for Lone Star Cement Corp. He has also been with Ideal Cement Co., Denver; Arkansas Portland Cement Co., Okay, Ark.; and Allis Chalmers Mfg. Co.

William F. Stowasser (*The Effect of Heat Treatment and Certain Additives on the Strength of Fired Magnetite Pellets*, P. 1223) is a photography enthusiast when not attending to the details of the workaday world. Born in Milwaukee, he graduated from University of Wis-

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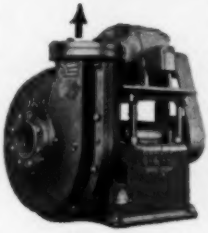
Lower Cost Pumping IS possible with DENVER SRL... Rubber Lined Pump



*Examples of DENVER SRL Rubber Lined Pump Curves**

Here's Why...

Power cost is 30% to 70% less than for other sand pumps on similar service. *REASON:* greater hydraulic efficiency resulting from simple design, rubber parts and lighter weight. Accuracy of rubber parts results in 1½ to 3 times greater efficiency than other sand pumps.

Learn More About Actual Savings

If you pump –¼" abrasives, describe your pumping requirements. Let us study and report specific advantages of Denver SRL Pumps over pumps you now use. Write Today!

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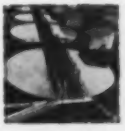
Denver Pulp Distributor

Size	Gals. of water per minute		20' head	40' head	60' head	80' head	100' head
2"x2" SRL	50	RPM	838	1090	1320	1525	
		HP	.60	1.5	2.8	3.2	
3"x3" SRL	100	RPM	760	1053	1303	1453	
		HP	1.1	1.9	3.4	4.3	
5"x5" SRL	300	RPM	590	800	956	1087	
		HP	2.4	5.4	8.3	11.5	
6"x6" SRL	1000	RPM		862	1005	1122	
		HP		14.4	22.6	30.0	
3"x3" SRL-C	150	RPM	870	1145	1385	1580	1745
		HP	1.5	3.2	5.3	7.2	9.6
5"x4" SRL-C	350	RPM	655	850	1020	1160	1280
		HP	2.9	5.4	8.3	11.4	14.5
8"x6" SRL-C	800	RPM	500	655	780	890	980
		HP	5.7	11.6	16.8	22.3	28.6
10"x8" SRL-C	2000	RPM	485	610	710	800	855
		HP	14.0	27.8	41.2	56.3	71.6

(*Multiply these horsepower ratings by the specific gravity of your pulp to determine actual break horsepower required.)



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Denver Reagent Feeders

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Dry Rolling Toggle

with Rockin' Chair Motion

A-1 Jaw Crushers

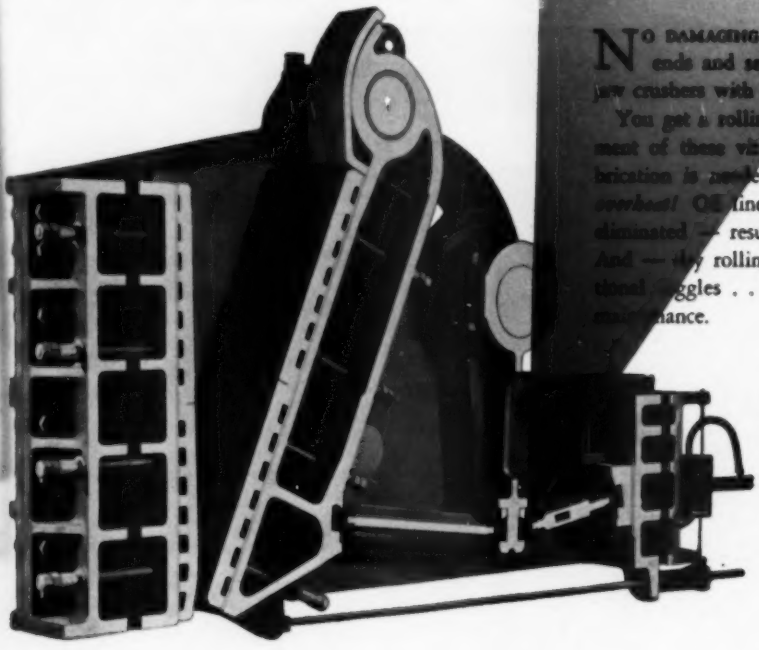
Reduces Maintenance and Downtime

No Lubrication Needed

NO DAMAGING sliding action of toggle ends and seats when you equip your jaw crushers with dry rolling toggles!

You get a rolling, "rocking chair" movement of these vital moving parts. No lubrication is needed, yet toggle ends *never overheat!* Oil lines and messy oil drip are eliminated — resulting in increased safety. And — dry rolling toggles outlast conventional toggles . . . operate with reduced maintenance.

A-3738



You can easily make your jaw crusher operations *more profitable* with dry rolling toggles. Call the A-C representative in your area, or write Allis-Chalmers, Milwaukee 1, Wis.

A-1 is an Allis-Chalmers trademark.

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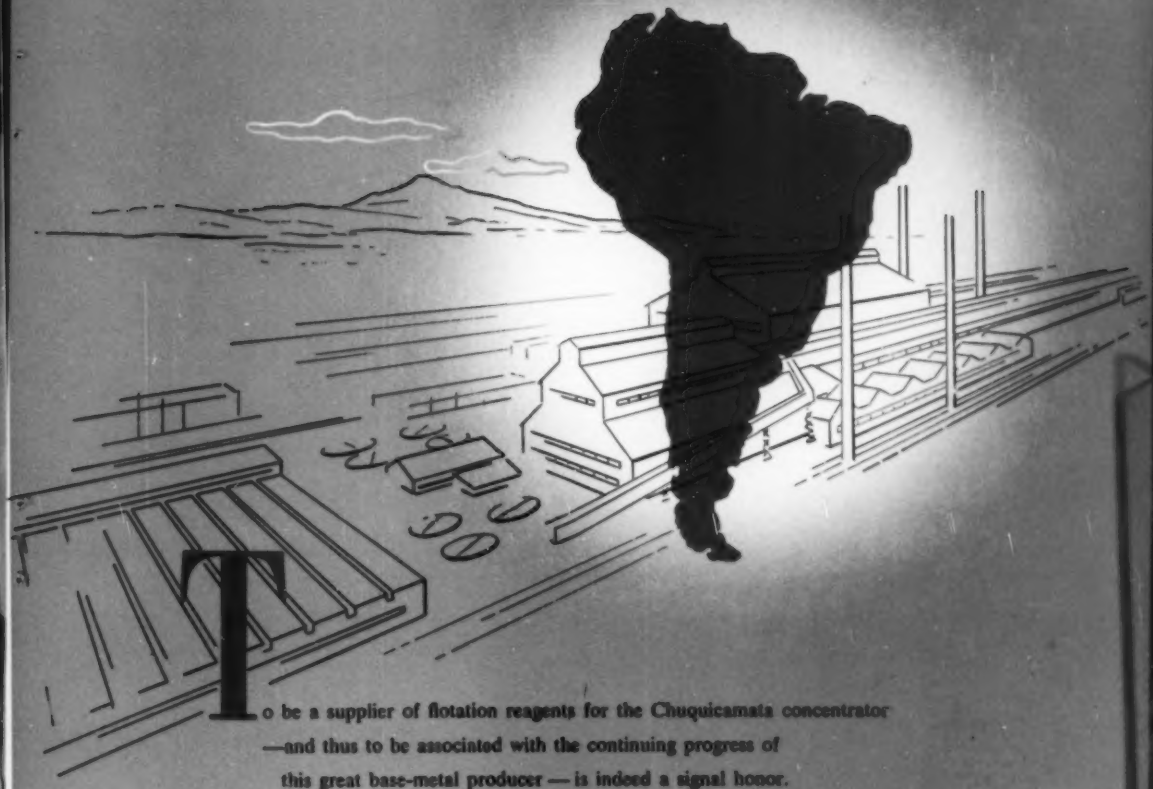

To be a supplier of flotation reagents for the Chuquicamata concentrator —and thus to be associated with the continuing progress of this great base-metal producer — is indeed a signal honor.

To be selected by Chile Exploration Company as a sequel to our long association with Andes Copper Mining Company is a privilege doubly valued by Cyanamid.

With the transition to sulphide ore at Chuquicamata, Cyanamid Flotation Reagents will be helping to concentrate a very substantial part of all the copper produced on the American continents.

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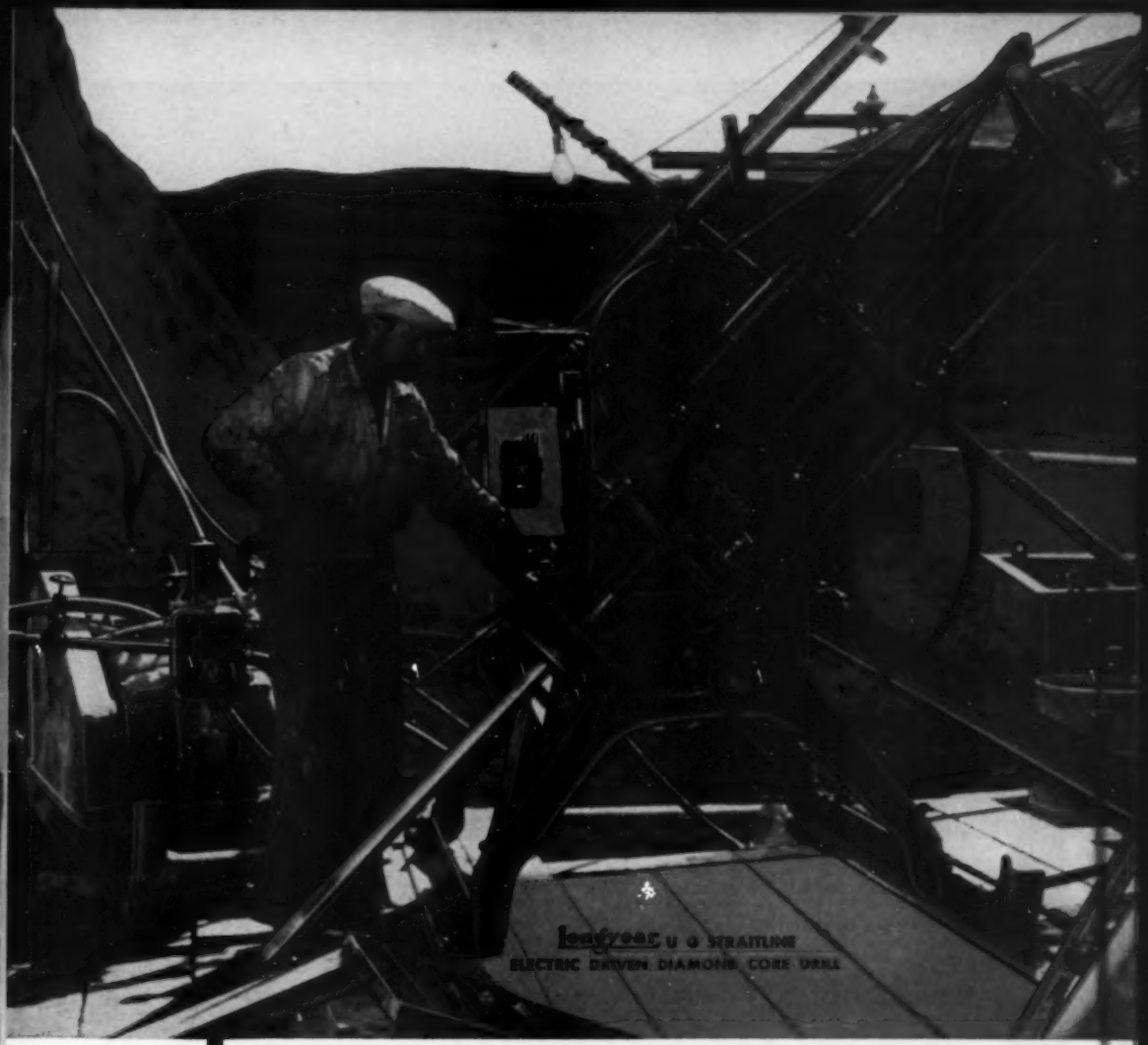

...Cores Tell The Story

Congratulations...

Chile Exploration Company and Anaconda Copper Mining Company

The E. J. Longyear Company compliments you on your outstanding mining achievement at Chuquicamata, Chile.

We are proud to have our electric driven drills used in this tremendous enterprise . . . the World's Greatest Open-Pit Copper Mine . . . with its most modern electrified operations.

To you go our best wishes for continued success!

E. J. LONGYEAR COMPANY

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CANADIAN LONGYEAR, LIMITED, NORTH BAY, ONTARIO, CANADA

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LASTS LONGER

FASTER TO INSTALL

COSTS LESS

...THAT'S WHY MINING MEN DEMAND *Carlon*

Hundreds of installations have proved that CARLON IS THE ANSWER TO MINE PIPING PROBLEMS. No other pipe gives you all the advantages of CARLON plastic pipe.

CARLON IS FASTER TO INSTALL . . . Flexible and furnished in long lengths . . . requires fewer fittings . . . makes-up faster, without special tools.

CARLON IS EASIER TO HANDLE . . . Weighs only 1/8th as much as metallic pipe . . . one man can handle 400-foot coil . . . no rigging equipment needed.

CARLON LASTS LONGER . . . Impervious to sulphurous waters, alkalies, acids, metallic salts and other corrosive wastes . . . **guaranteed** against rot, rust and electrolytic corrosion . . . has served over four years without maintenance under extremely severe mine conditions.

CARLON COSTS LESS . . . Cuts installation costs to minimum and last many times longer than ordinary pipe, yet initial price is about the same.

To be sure you get the best mine pipe available today, demand CARLON . . . SPECIFY THE PIPE WITH THE STRIPE!

NOM. SIZE	O.D.	I.D.	EST. BURST P.S.I.	WT. LBS. PER FT.	SHPD. LENGTHS
½"	0.840	0.622	340	0.103	400 ft. coils
¾"	1.050	0.824	320	0.140	400 ft. coils
1"	1.310	1.070	300	0.181	300 ft. coils
1¼"	1.660	1.380	300	0.267	300 ft. coils
1½"	1.900	1.610	300	0.320	250 ft. coils
2"	2.378	2.070	170	0.445	200 ft. coils
2½"	2.875	2.469	170	0.680	200 ft. coils
3"	3.504	3.070	165	0.910	100 ft. coils
4"	4.504	4.030	130	1.250	25 ft. str.
6"	6.630	6.070	115	2.230	25 ft. str.

ALL CARLON PIPE AND FITTINGS ARE FACTORY TESTED FOR MORE THAN 8 HOURS AT GREATER THAN WORKING PRESSURES.

Write for Catalogs.

TM-CP

CARLON PRODUCTS CORPORATION

PIONEERS IN PLASTIC PIPE!

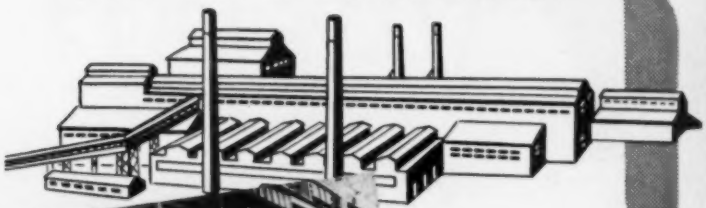
In Canada: MICRO PLASTICS, Ltd., Acton, Ontario

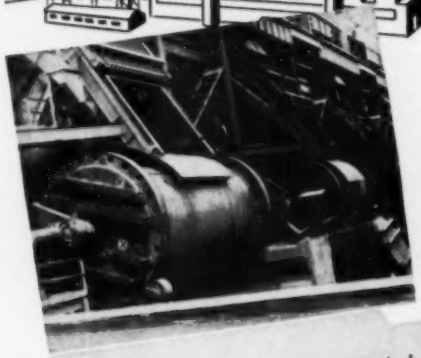
10300 MEECH AVENUE • CLEVELAND 5, OHIO

Traylor

MACHINERY GOES TO CHUQUICAMATA!

- FOUR 13'-0" x 30'-0" Pierce-Smith Converters
- ONE 41'-3' Anode Casting Wheel
- ONE 10" x 16" Type H Jaw Crusher
- ONE 7'-6" x 315'-0" Rotary Kiln
- ONE 60" Gyratory Crusher





SITE of the world's largest known copper ore body, "Chuqui" is the home of the already famous 111 million dollar sulphide plant newly built by the Chile Exploration Co. Many of the world's best known equipment manufacturers have sent their products to this great new enterprise. Among them is Traylor, known the world over for the dependable operation of its mining machinery.

For many years, Traylor has supplied the world's leading copper and nickel producers with converters of the Pierce-Smith type in sizes up to 13' x 35'.

Photo shows the new plant going up around a 7'-6" x 315' Traylor Rotary Kiln. This Kiln was specially designed for the job.

Plant's 30,000 Ton Daily Capacity Requires DEPENDABLE Machinery

For 50 years, Traylor engineers have worked with the mining industry, studying its needs and developing advanced equipment to meet its demands. That is why, wherever extreme efficiency is required, Traylor machinery is specified. It takes experience to keep machine designs ahead of production demands. Traylor has experience . . . half a century of it.

Traylor **ENGINEERING & MANUFACTURING CO.**
1434 MILL ST., ALLENTOWN, PA.

SALES OFFICES: New York • Chicago • San Francisco
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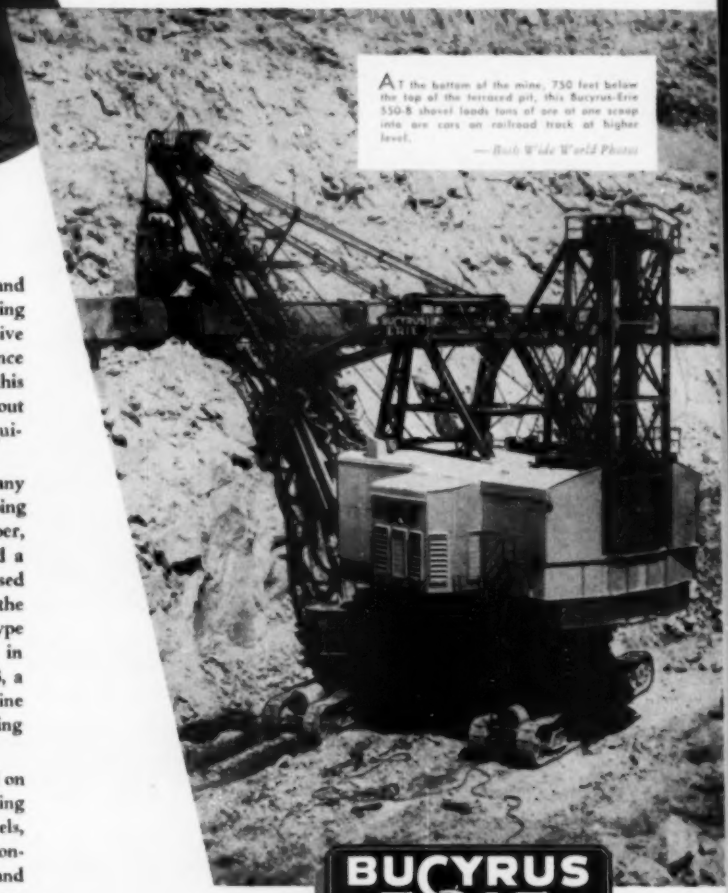
Fill in . . . . mail coupon for eight page catalog showing Traylor's "Equipment for the Mining Industry".

Name: _____
Position: _____
Company: _____
Address: _____
State: _____

Here's Proof of Dependable

HIGH OUTPUT

THE Chuquicamata orebody — the largest known copper ore deposit in the world — has been worked by open-pit methods for 38 years.



AT the bottom of the mine, 750 feet below the top of the terraced pit, this Bucyrus-Erie 550-B shovel loads tons of ore at one scoop into ore cars on railroad track at higher level.

— *Acme Wide World Photos*

In expanding its copper mining operations and searching for more economical means of uncovering ore, Chile Exploration Company has made extensive use of Bucyrus-Erie mining shovels — for instance the 550-B shown here. In the past 18 months, this machine, with its 11-cu. yd. dipper, has loaded about 17½% of all ore removed daily from the Chuquicamata pit.

The 550-B, on the job since 1946, is one of many Bucyrus-Eries used at Chuquicamata since stripping operations began on old Bench "C" in September, 1914. Seven 95-ton Bucyrus steam shovels and a 75-ton steam-driven Bucyrus locomotive crane, used earlier in building the Panama Canal, started the job. Two years later the first of nine modern-type Bucyrus railroad shovels was purchased. And, in 1925, Chilex bought its first Bucyrus-Erie 120-B, a 5-cu. yd. mining shovel. Today, there are nine 120-B's and the 550-B doing stripping and loading in the Chuquicamata pit.

Mining companies all over the world depend on Bucyrus-Erie shovels to handle digging and loading assignments. They know that Bucyrus-Erie shovels, with their fast operating cycles and durable construction, provide the dependable high-output and low-cost performance that means profitable operation.

1191S2C

BUCYRUS ERIE

SOUTH MILWAUKEE, WISCONSIN

Equipment From Frantz TRACTOR COMPANY

Allis-Chalmers tractor from Frantz leads the convoy at 'Chuqui'.

Helps Speed Construction at "Chuqui"!

Tracto-Shovel from Frantz leads slag on conveyor at 'Chuqui'.

FRANTZ is proud to have furnished 28 units of construction equipment to Chile Exploration Co. for work on the new sulphide plant and open-pit copper mine at Chuquicamata. To companies like Chile Exploration, having central buying offices in New York City, FRANTZ offers complete lines of all types of construction equipment *plus* the facilities to keep up a steady flow of parts and maintenance items to jobs of any size — anywhere! . . . Let FRANTZ show you how you can centralize *all* your equipment and parts purchasing.

FRANTZ TRACTOR COMPANY

45 ROCKEFELLER PLAZA, SUITE 3661, NEW YORK 20, N. Y.

NEW YORK DISTRIBUTOR FOR ALL TYPES OF CONSTRUCTION EQUIPMENT

Manufacturers News

New Products • FILL OUT THE COUPON FOR MORE INFORMATION • Equipment

Servicing School

Opening of a special factory training course in servicing earthmoving equipment is announced by the *Euclid Road Machinery Co.* Four

beginning and four advanced one week training courses will be offered to representatives of distributors and customers. **Circle No. 1**

Core Drill

The new *Acker* "Teredo" core drill, is named after the "boringest little animal in the world" to underline its function as a light, compact, portable core drill capable of depths to 600 ft. While light enough to be truck or jeep-mounted, it offers the combination of a three-speed totally enclosed transmission with either hydraulic or screw feed drill heads. Full 24-in. travel gives maximum footage per shift, 360° rotation permits freedom of hole location, a transmission mounted cathead winch is used in pulling rods and casings, and a power take-off is provided for a pump. **Circle No. 2**

Plastic Pipe

Planned production of both flexible and semi-rigid plastic pipe is announced by *Triangle Conduit & Cable Co.* Rigid and high-impact pipe will be added to the line later. Three types: non-toxic, industrial, and suction, are being made and in sizes from ½ to 6 in. **Circle No. 3**

Bin Level Control

Fly ash, collected in a Cottrell precipitator at a Midwestern public utility power plant, is automatically measured and the bin level controlled by a single model 40 Rotax controller, product of the *Foxboro Co.* A temperature bulb inside the precipitator senses when dust shields it from the higher temperature of the flue gases. The system is applicable for bulk materials where the bin temperature is above or below ambient temperature. **Circle No. 4**

Flotation Machine

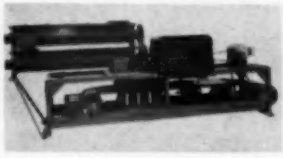
Jetair flotation machines exhibited by *Morse Bros. Machinery Co.*, at the AMC Denver Mining Show incorporated a new boltless impeller housing that gives immediate access to the impeller for inspection and replacement. They also displayed for the first time a plastic model Concen-Tank thickener. **Circle No. 5**

Controller

The Pneumatrol P-4 pneumatic controller for rate of flow, differential pressure, liquid level, viscosity, pressure, and temperature applications, designed by *Fischer & Porter Co.* to use a true motion control action, is available for all types of control coupling. **Circle No. 6**

Transportoweigher

Sintering Machinery Corp. has designed a new feed regulating unit for automatically and continuously transporting, weighing, and maintaining pre-set feed rate for finely divided material. The photograph



shows a 24 in. Transportoweigher to handle ground phosphate rock at capacities between 10 and 24 tons per hr. An integrator totalizes tonnage passing over the belt while the tonnage indicator gives the instantaneous weighing rate. To eliminate skirt board drag the material is fed onto the belt through two screw conveyors. **Circle No. 7**

Trenchmobile

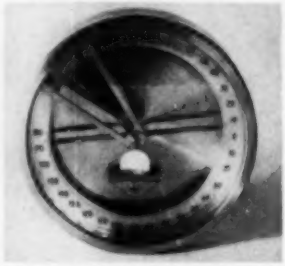
Recently introduced by its maker, *Koehring Co.*, the Trenchmobile is a highly mobile, rubber tired trench digger for utility trenching assignments. Widths of 8 or 12 in. are dug at feeds up to a maximum of 20 fpm, and depths to 5 ft. The unit can be driven from one job to the next at speeds of 12 mph. **Circle No. 8**

Dockboard

Bridging big differences between truck and dock heights the lightweight *Magnesium Co. of America* ramp-dockboard, makes loading and unloading safer and faster. In two sections, the ramp and dockboard are locked together by pins, allowing rise and fall with the truck floor during loading. **Circle No. 9**

Dip Needles

Available for the first time in the United States are the model D-2 dip needle and a self orienting dip



needle distributed by the *Radiac Co.* The model D-2 gives swing readings with an instrument built for rugged use, and the self orienting model is designed for fast traversing where disturbance is slight. **Circle No. 10**

Prestressing

Stressteel Corp. has been formed to produce large diameter, high strength steel tensioning units for prestressing concrete. Bars ½ to 1⅛ in. diam and up to 80-ft length, together with their end anchorages were developed recently in England. *Stressteel* has been established to manufacture and sell the bars in the Western Hemisphere. **Circle No. 11**

Jalloy Steel

Cutting edges of the bucket in the photograph were made of Jalloy, a special alloy steel, exclusive product of *Jones & Laughlin Steel Corp.* The bucket is on a Strait-Line hy-

draulic loader equipped tractor working in a Maine sand and gravel pit. Jalloy reportedly is easy to weld using regular ferritic-coated rod and is widely used for abrasion and impact resistance. **Circle No. 12**

Pipe Wrenches

Redesign of the *Ampco Metal Co.* pipe wrench enables users to replace worn gripping surfaces with jaw inserts at about one tenth of previous cost. Existing wrenches can be modernized with a new upper jaw assembly. **Circle No. 13**

Free Literature

1818-FT CONVEYOR BELT, making a roll almost 12-ft diam, presented an unusual problem for Goodyear Tire and Rubber Co.'s shipping department. Believed to be the longest conveyor belt ever shipped in one continuous piece, it required special bridge and underpass clearance permits from Ohio and Indiana for highway transportation.

(14) SCREENS: Description of *Allis-Chalmers'* complete line of vibrating screens for the mining industry is contained in a new 8-page bulletin. Primary scalping, secondary scalping, and conventional screens for wet or dry sizing and screens for heavy media separation are included. Also given is a typical flowsheet showing how horizontally operated Low-Head screens are used in the sink and float process.

(15) MILLS: Forty-six years of experience and 4000 installations throughout the world lie behind the seven basic types of mills illustrated in a recent bulletin from the *Hardinge Co.* Fields of application and materials handled in various installations are listed.

(16) BLOWERS & COMPRESSORS: A brochure describing the manufacturing facilities and the axial and centrifugal blowers and compressors made by the *Sawyer Bailey Corp.* is now available.

(17) LINATEX: The first catalog of the *Linatex Corp. of America* has been issued, outlining the advantages claimed for this special form of natural rubber, when used in corrosion, abrasion, and vibration resisting applications.

(18) WIRE ROPE: Twenty-four pages packed with concentrated wire rope data is the way *E. H. Edwards Co.* describes a booklet which they are now making available. It was condensed from the Edwards general catalog to provide users with a concise guide to every important phase of wire rope production, maintenance, and use.

(19) WELDING: "Amsco Means Control of Impact and Abrasive Wear" is the title of an *American Brake Shoe Co.* publication on manganese steel uses in materials handling. Another publication outlines application of welding rods for rebuilding and hard facing. Bulletin WA-77 lists repointers, grouser bars, filler bars and plate for maintaining and rebuilding heavy earth moving equipment, shovels and scrapers and bulletin 650-W describes Amsco welding products for hard facing and repair.

(20) COALMASTER DRILL: After several years of design, laboratory testing, and extensive field testing *Central Mine Equipment Co.* has started commercial production of its

new drill head known as the Coalmaster ROCKet Head. Using nine Expenda bits in the arrangement shown in the photograph, the new design is claimed to increase drilling speed, reduce driving power required, and be lower in first cost.

(21) CREOSOTE: "Why Creosote Protects Wood Today Every Bit as Well as Fifty Years Ago," has been issued by the *Koppers Co.*

(22) FLOTATION MACHINES: *Denver Equipment Co.'s* new Super Rougher flotation machine is in

quantity production. A bulletin now available describes its features for roughing and scavenger flotation.

(23) MINE CONVEYORS: *Hewitt-Robbins Inc.* describes their line of mine conveyors in a 16 page booklet, which emphasizes that they are the one company in the world to assume complete responsibility for both elements of a mine conveyor—machinery and belt. Robbins mine conveyors are available in widths of 24 to 36 in. or wider, and with several types of idlers and drives.

(24) DIESELS: Publication of 12 two-color, two-page specification sheets on *Nordberg Mfg. Co.* type 4FS one, two and three cylinder diesel engines is announced. These bulletins give complete engine specifications, and outline drawings of the various models. These engines are of the heavy-duty, four-cycle type, with 4½ in. bore and 5¼ in. stroke, ranging in power from 10 to 45 hp.

(25) FRACTIONAL MOTORS: Detailed information on 35 Emc and Cyclohm fractional motors in ratings from 1/2000 to ½ hp is provided on a unique slide rule offered by *Howard Industries, Inc.*

Mining Engineering **December**
29 West 39th St.
New York 18, N. Y.

Please send me { More Information ☐ / Free Literature ☐ / Price Data ☐ } on items indicated

1	2	3	4	5	6	7	8	9	10
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Students are requested to write direct to the manufacturer.

Name_____ Title_____

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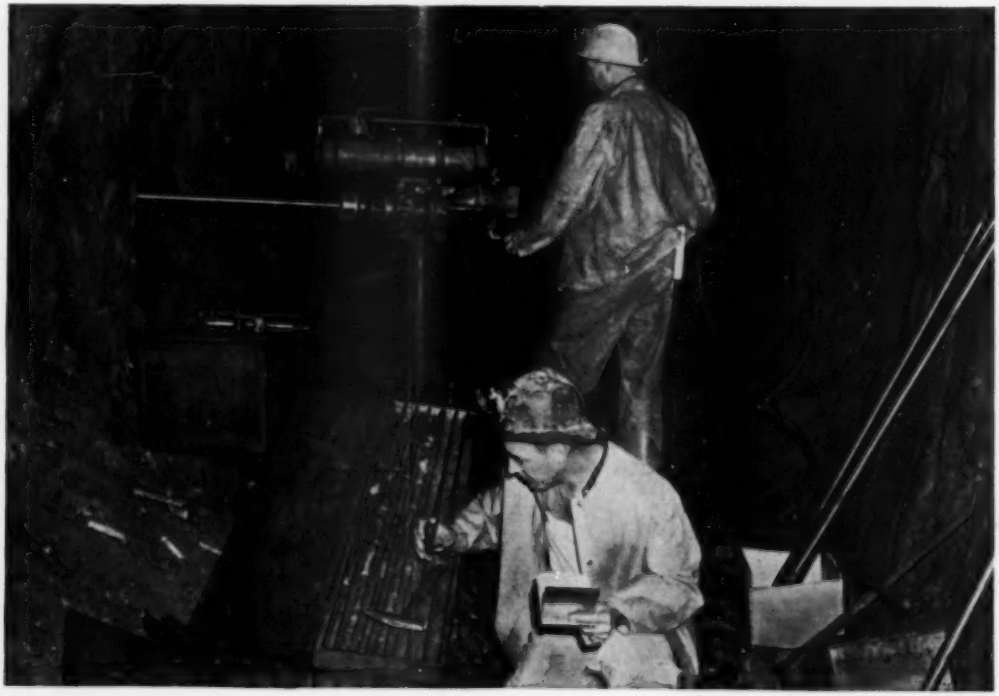
The compact design of CP ROTORdrifters — with rotary feed motor built into the drill between backhead and drifter cylinder — makes them particularly suitable for close-quarter operations. CP MOTORdrifters — with feed motor on the end of the shell — are specifically designed for the longer feeds sometimes preferred in large bore tunnels.

Every type of ROCK DRILL

for modern mining

Designed both for blast hole and exploratory drilling, the air-driven CP No. 55 DIAMOND CORE DRILL has the most powerful vane-type air motor ever built into a diamond drill. Conservatively rated at 500 feet with E Rods and EX Fittings.



CP-34 STOPERS have all the features that assure maximum footage from tungsten-carbide bits. The combination of strong rotation, fast hitting action, and the right foot-pound blow give maximum penetration and long bit life. Medium weight, fully cushioned piston and fine running balance make the CP-34 easy to handle. Constant air blow keeps all sludge out of the drill.

For the toughest drilling jobs, the heavier CP-44 STOPERS are recommended.

The complete line of CP SINKER DRILLS, ranging from the CP-22 (28-pound) to the CP-50N (102-pound), includes the CP-32, a top performer in the 45-pound class, and the CP-59, leader in the 55-pound class. Both of these can be furnished with the CP AIRLEG, which provides a rigid, well-balanced support and effectively absorbs all recoil.

Designed for maximum speed and efficiency in tunnel driving, the rail-mounted G-600 AIR ACTUATED DRILL JUMBO is furnished in single and multi-boom units.

OTHER CP MINING EQUIPMENT

Exploratory Diamond Drills, gasoline, air and electric powered . . . Wagon Drills . . . Sludge Pumps . . . Sump Pumps . . . Pneumatic Impact Wrenches . . . Stationary and Portable Compressors . . . Pneumatic and Electric Construction and Maintenance Tools.

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PNEUMATIC TOOLS • AIR COMPRESSORS • ELECTRIC TOOLS • DIESEL ENGINES
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AT CHUQUI...

Dorr Equipment and Methods provide better metallurgy at the mill and modern sewage treatment for the community

For tailings thickening...

Four 300′ dia. Dorr Traction Thickeners

For concentrates thickening...

Three 130′ dia. Dorr Torq Thickeners

For middlings thickening...

Five 80′ dia. Dorr Torq Thickeners

For regrinding middlings...

Five 12′ x 28′ 4″ Dorr Quadruplex Classifiers

For sewage treatment...

A complete Dorr Biofiltration System with a 1,875,000 GPD capacity for a design population of 22,500. Equipment includes two 85′ dia. Dorr Clarifiers, one 105′ dia. Dorrco Distributor, and an 85′ dia. Dorr Digester.

For milk of lime preparation...

One 4′ x 25′ Dorr Classifier
One 32′ dia. Dorr Agitator

AND THROUGH OUR EXCLUSIVE REPRESENTATION:

For dewatering copper concentrates...

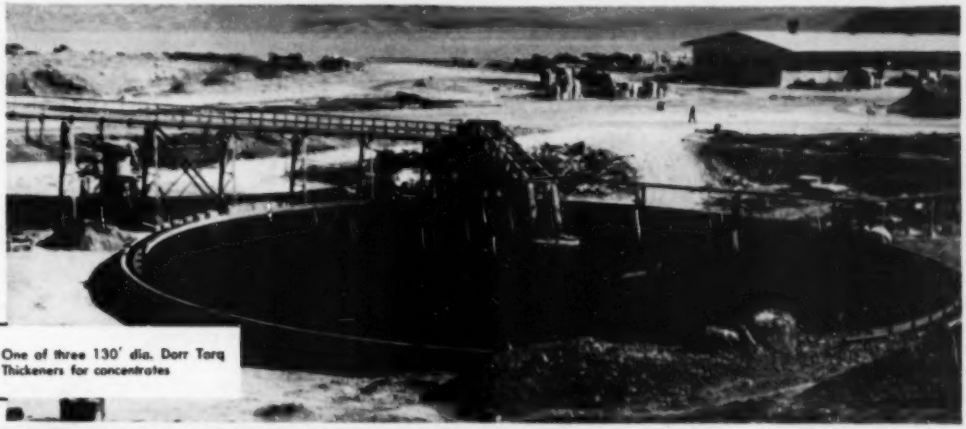
Four American Filters each 12′ 6″ dia. x 7 disc

For agitation in dechloridizing plant...

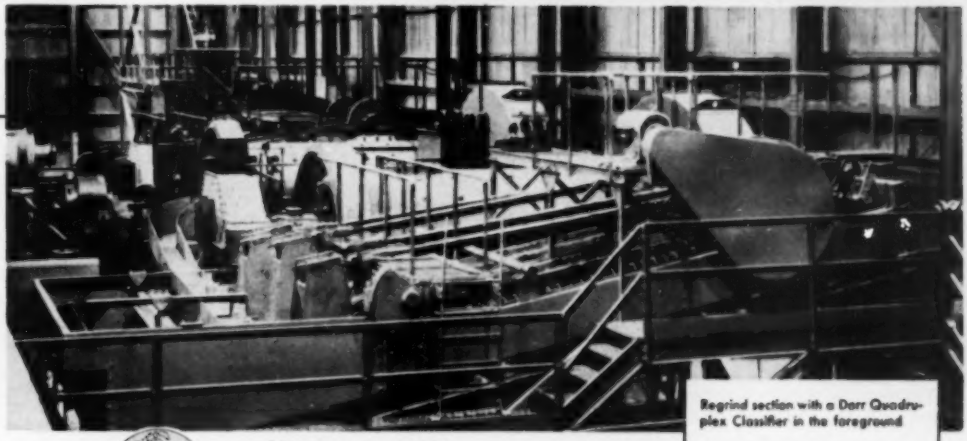
Five Lightning Top Entering Agitators

Torq is a trademark of The Dorr Co. Reg. U. S. Pat. Off.

One of the four 300' dia. Dorr Traction Thickeners handling tailings



One of three 130' dia. Dorr Torq Thickeners for concentrates



Regrind section with a Dorr Quadruplex Classifier in the foreground

Better tools TODAY to meet Tomorrow's demand

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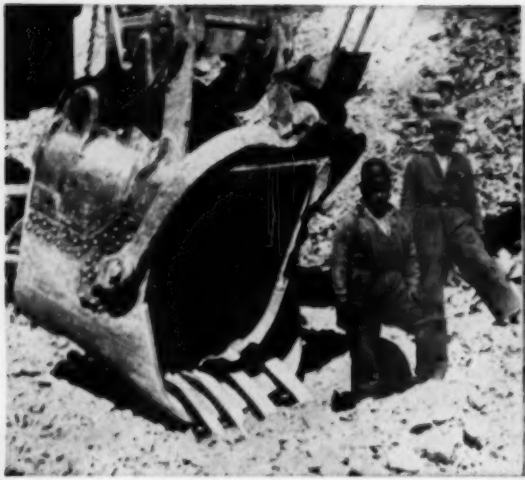


CHILE EXPLORATION COMPANY DEPENDS ON AMSCO DIPPERS

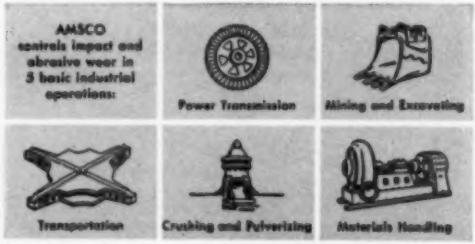
It's a long way to Chile from the AMSCO Plant in Chicago Heights, Illinois. And it's even farther when difficulties in shipping to remote Chilean diggings are considered. That's one reason why mining engineers in that part of the world specify dependable long-wearing AMSCO Manganese Steel dippers.

Made *entirely* of Manganese Steel, "The Toughest Steel Known," AMSCO dippers actually work-harden under severe impact and abrasion. Acquiring a high polish that resists abrasive wearing action, and surface hardening to 450-550 Brinell, Amsco Manganese Steel Dippers retain all their inherent body metal strength and toughness.

AMSCO Dippers are not only made of Manganese Steel, but are engineer-designed for maximum materials handling efficiency:

- **One-piece Manganese Steel back casting.**
- **Top and bottom have cast-in, double-wall box sections for maximum strength, minimum weight.**
- **Cast-in pin connecting lugs for handle, back, braces, bail.**
- **Bail is a one-piece casting with elongated tubular cross section for strength and lightness.**
- **Teeth are set for clean, fast digging.**
- **Interior is smooth and slightly tapered for fast dumping.**

AMSCO All-Manganese Steel Dippers are available in two-piece welded, three-piece renewable lip, and Missabi designs—or can be adapted to your unusual materials handling specifications. Available in sizes to fit all power shovels. Contact your shovel manufacturer about AMSCO Dippers, or write AMSCO, Chicago Heights, Illinois.



AMERICAN Brake Shoe COMPANY

AMERICAN MANGANESE STEEL DIVISION

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Other Plants: New Castle, Del., Denver, Oakland, Cal., Los Angeles, St. Louis. In Canada: Joliette Steel Division, Joliette, Que. Amsco Welding Products distributed in Canada by Canadian Liquid Air Co., Ltd.

On this important enterprise are Allis-Chalmers tractor shovels — ideal for cleanup work, loading slag on conveyor belt for slag melting furnace, and other loading and hauling work.

Here an Allis-Chalmers torque converter tractor, one of a fleet of A-C machines, excavates a pipe trench at Chuquicamata, Chile, site of the new open-pit copper mine and sulphide plant of Chile Exploration Co., a subsidiary of Anaconda Copper Mining Co.

FLEETS OF ALLIS-CHALMERS TORQUE CONVERTER TRACTORS BRING

High Output with Low Upkeep

TO CHILE EXPLORATION CO. GIGANTIC "CHUQUI" PLANT JOB

Here's another big job where Allis-Chalmers torque converter tractors are setting new performance standards, new handling ease and — most important — new productivity to boost output and cut costs. Here's why —

HYDRAULIC TORQUE CONVERTER DRIVE gives the operator the equivalent of hundreds of gear ratios in two speed ranges . . . automatically selects the proper speed for maximum production on all types of work. With shifting virtually eliminated, operator can concentrate on his work load.

TORQUE CONVERTER DRIVE takes full advantage of available horsepower for rolling bigger loads and tackling tougher jobs. Master clutch engages under cushioned protection and power is delivered instantly with velvet-like smoothness. It's easier on the whole tractor, means fewer repairs, longer life for tractor and auxiliary equipment.

TORQUE CONVERTER DRIVE actually multiplies torque up to four and one-half times, develops tremendous drawbar pull to start loads easily, smoothly. The operator can quickly synchronize tractor's speed with drawn or pulled equipment — an advantage that reduces wear and tear and equipment damage.

You can get this same kind of productive power from Allis-Chalmers HD-20 torque converter tractors on your jobs. Get the full story from your Allis-Chalmers dealer soon — see why the HD-20 does more work, more easily, at less cost.

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TRACTOR DIVISION — MILWAUKEE 1, U. S. A.

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Now at

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(CHILE, S. A.)

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COPPER-MOLYBDENUM-ALLOY

Grinding Balls

Used and PROVED Around The World

Deliver Higher Efficiency and Economy

THE STORY of Chuquicamata can be told only in superlatives . . .

THE LARGEST known copper orebody in the world, will produce more than 300 million pounds of blister copper a year, and treat 30,000 tons of ore a day. The Anaconda workings are a landmark in mining operations.

SHEFFIELD MOLY-COP GRINDING BALLS play a vital role in this record-breaking project. Their longer service in the new sulphide concentrator means lower grinding costs, fewer chargings, and less "down time."

SUPERLATIVE PERFORMANCE of Sheffield Moly-Cop grinding balls is the result of years of research, and quality-controlled methods of manufacture—from initial alloy steel production to final heat treatment.

LET SHEFFIELD prove how Moly-Cop Grinding Balls will do a better job for you.

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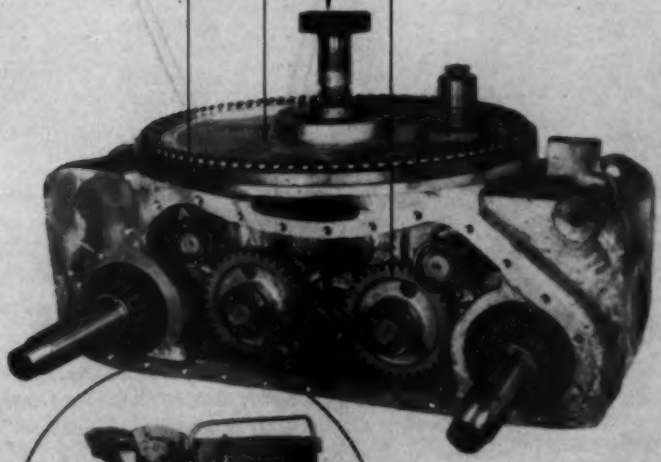
WHAT ALLOY STEEL CASTINGS* MEAN IN UNDERGROUND LOADING MACHINES

The structure of steel castings is uniform for greater strength, shock and stress resistance.

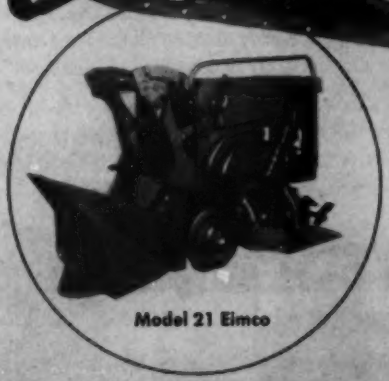
In steel castings the metal is distributed where it will do the most good—for maximum strength and minimum weight.

High rigidity, minimum deflection, accurate alignment, close tolerance and better fit are all inherent characteristics of steel castings.

Steel castings have high fatigue resistance, maximum endurance and longer life—ideal qualities for critically stressed parts.



Model 21 Eimco Lower Deck



Model 21 Eimco

***All EIMCO Loaders are built with alloy steel parts throughout. This is an exclusive EIMCO feature.**

EIMCO

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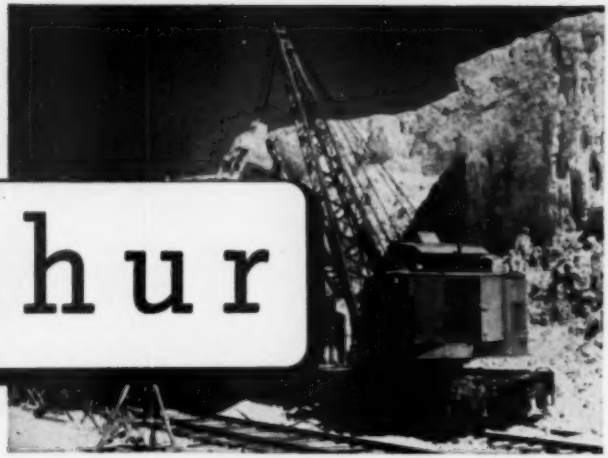


Sulphur

Thousands of tons mined daily, but where does it all go?

Did you ever have the misfortune on a steaming, sticky, sultry day to sit it out on a crowded parkway, bumper to bumper, waiting for traffic to clear? No doubt your thoughts were plenty sulphurous but probably not along the lines we have in mind.

We're thinking of the mineral Sulphur and its link with the automobile. Each car accounts for a substantial poundage of Sulphur, some estimates put it at around 25 pounds for the average car. Give or take 5 pounds, it shows that a tremendous tonnage of Sulphur is needed each year to put cars, buses and trucks on the road ready to operate. And don't forget the tire and battery replacements going on every day.

Sulphur enters the automobile picture through the tires, steel sheets, plated and plastic fittings, glass, battery acid and parts, copper tubing and wiring . . . all of which call for the use of Sulphur or its compounds in connection with their manufacture.

Can you wonder that Sulphur goes into industry just about as fast as the sulphur producers of the Gulf Coast Region can get it above ground and cooled preparatory to shipment?

Texas Gulf Sulphur Co.

75 East 45th Street, New York 17, N. Y.

Sulphur Producing Units: Newgulf, Texas • Moss Bluff, Texas
Spindletop, Texas • Worland, Wyoming

WIRE ROPE, too, resists attack with the RIGHT KIND of muscle

Brawny chest and shoulder muscles make the gorilla a formidable foe in the eternal warfare of the jungle. Defending himself and his family, this 450-pound heavyweight stands erect and fights back with sledge-hammer blows of his mighty forearms.

In wire rope, too, it takes the right kind of muscle to resist constant attack of abrasion, corrosion, bending fatigue, load strain and shock stress.

Complete quality control of Wickwire Rope means that you can always count on the right grade of steel and size of wire; the right construction and lay of the rope for best results on your particular job.

See your Wickwire Rope distributor or contact our nearest sales office.

THE COLORADO FUEL AND IRON CORPORATION — Abilene (Tex.) • Denver • Houston • Odessa (Tex.) • Phoenix • Salt Lake City • Tulsa
THE CALIFORNIA WIRE CLOTH CORPORATION — Los Angeles • Oakland • Portland • San Francisco • Seattle • Spokane
WICKWIRE SPENCER STEEL DIVISION — Boston • Buffalo • Chattanooga • Chicago • Detroit • Emlenton (Pa.) • New York • Philadelphia

1001

A YELLOW TRIANGLE ON THE REEL IDENTIFIES WICKWIRE ROPE

WICKWIRE ROPE

CF&I

PRODUCT OF WICKWIRE SPENCER STEEL DIVISION
THE COLORADO FUEL AND IRON CORPORATION

GIANT NICKEL STEEL SHAFT... installed as a replacement on a 54" crusher, to assure maximum service life for The International Nickel Co., at Copper Cliff, Canada. Overall length 21'6", maximum diameter 3'8", bore 6".

How Nickel Helps a Crusher

PUT THE SQUEEZE ON COSTS

Many forgings are so large that only part of the mass can be worked under the press before the steel has to be reheated. These large sections of steel, typified by this crusher shaft, so limit the cooling rate as to make liquid quenching ineffective.

Consequently, improved strength, hardness and other properties that prolong life of large forgings are much more dependent upon wise selection of alloy content than is the case with small forgings.

Because of these facts, the large crusher shaft shown above was forged from a 160,000-pound ingot of 2¾ per cent nickel steel... produced, rough-turned and heat-treated by the Bethlehem Steel Company, and finish-machined by the Traylor Engineering Company of Allentown, Pa.

After normalizing and tempering, two tests on longitudinal specimens, taken from a prolongation at mid-radius, averaged as follows:

Tensile Strength	80,000 p.s.i.
Yield Strength	51,000 p.s.i.
Elong. in 2"	28.0%
Red. of Area	58.3%

The strengthening effect of nickel on ferrite is independent of carbon content or heat treatment of the steel, and its effectiveness in reducing the rate and temperature of the upper transformation, induces better response to the necessarily milder heat treatments used.

Nickel alloy steels may help you obtain peak performance from vital parts of *your* products or equipment. Send us the details of your problems for our suggestions. Write us now.

At present, most of the nickel produced is being diverted to defense. Through application to the appropriate authorities, nickel is obtainable for the production of engineering alloy steels for many end uses in defense and defense supporting industries.

INCO

THE INTERNATIONAL NICKEL COMPANY, INC. 67 WALL STREET NEW YORK 5, N. Y.

Claims of 16 gold mining companies seeking damages from L-208 will be heard in Court of Claims starting Jan. 26. Earlier this year, judges ruled that 3 gold mining companies could sue for damages from the wartime order closing their mines. Congress later waived the statute of limitations thus permitting other gold miners to get in their claims.

Despite power shortages, third quarter production of aluminum exceeded that of the previous period by 2.2 pct. However, September production was 16 million lb less than in August.

Hewitt-Robbins, Inc. dedicated a new $1 million addition to the company's Buffalo, N. Y. belt conveyor plant. New equipment, including a $220,000 press, will make belts up to 70 in. wide, compared with the former limit of 60 in.

Charges of a tin cartel in Malaya are not backed up in the latest U. S. Government report according to the Malayan Tin Bureau. The U. S. mission reported that it "did not discover any evidence of curtailment of tin output by the mining and smelting industries." It noted that guerilla activity had halted exploration and expansion attempts by producers.

Latin American steel production set a new record in 1951 with 1.6 million tons. Brazil led with 900,000 tons, Mexico was second with 390,000 tons, and Chile produced 200,000 tons.

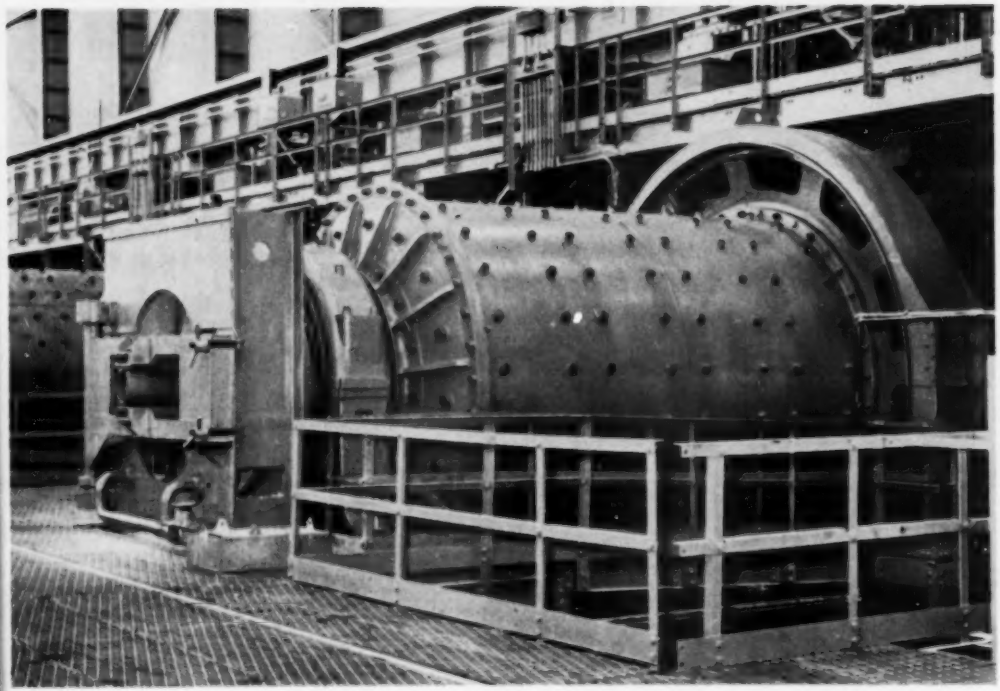
First iron ore shipments ever made by the Dominican Republic reached Chester, Pa., for the Chester Blast Furnace Co. The shipment was 5450 tons, but it is expected that some 10,000 tons will reach the U. S. monthly. U. S. Steel Corp., is reported to have purchased small amounts for testing purposes.

Full extent of potash discoveries in Saskatchewan are uncertain, but exploratory drilling revealed that they may extend over an area of 250 sq miles and in thicknesses up to 11 ft. The discovery was made about 150 miles northwest of Saskatoon, at Vera and Unity. Western Potash Co., has spent about $1 million for exploratory drilling. A 3450-ft shaft has been sunk.

While the operator sits on the surface controlling the machine by push button, a new continuous coal mining machine underground turns out up to 1200 tons per day. Developed by Union Carbide & Carbon Co., the electrically-powered machine moves on tractor treads and has 4 intermeshing rotary cutting heads at the front. Coal is brought out on conveyors.

Egypt plans to build its own steel industry, using electrical power generated from the Nile River, and iron ore from upper Egypt. Initial output will be approximately 150,000 metric tons. Construction bids will be taken internationally.

at "Chuqui" too,



The following M & S and CIW equipment is being used in this new installation:

10—10′ 8″ x 14′ Marcy Open End Rod Mills in open circuit for primary fine crushing, producing feed for regrind mills.

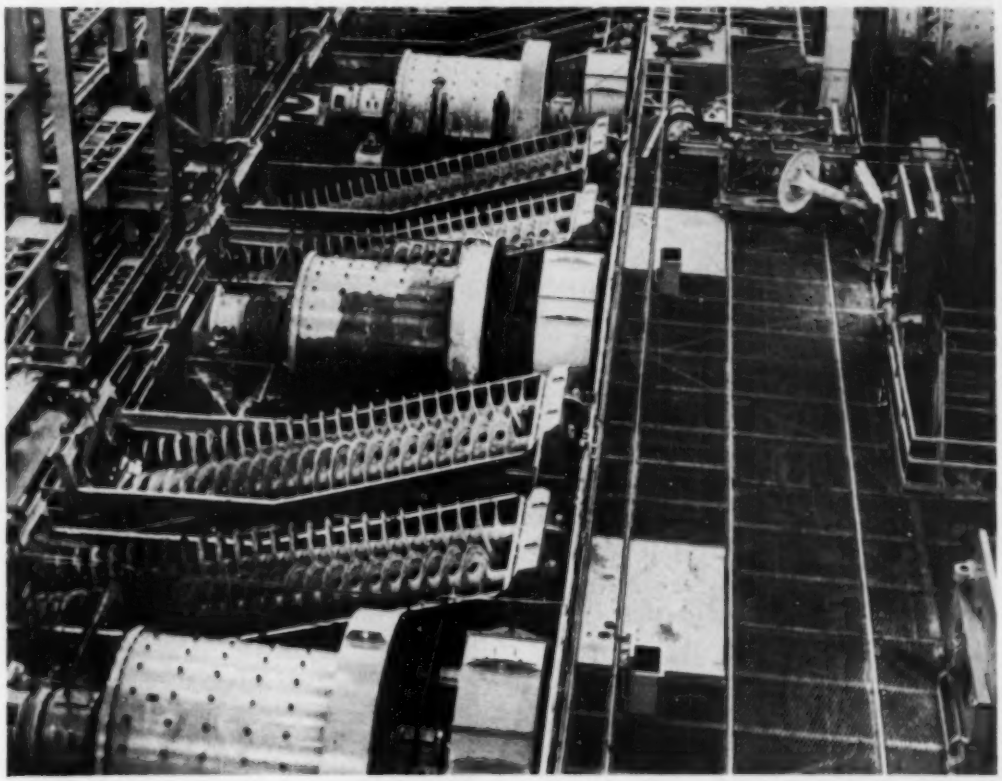
10—10′ x 12′ Marcy Regrind Ball Mills in closed circuit with 20—54″ Akins Duplex Submerged Spiral Classifiers.

6—6′ 7″ x 12′ Marcy Ball Mills operating as concentrate regrind mills and lime mills.

it's

MARCY MILLS AND AKINS CLASSIFIERS

Like many other successful mining companies, Chile Exploration Company relies on Marcy Mills and Akins Classifiers for dependable mechanical operation and efficient, low cost grinding and classification.



Let our engineers show you the reason why so many companies choose Marcy and Akins... proved results from all kinds of grinding and classification installations.

The Mine & Smelter Supply Co.

Denver 17, Colorado

OFFICES IN SALT LAKE CITY, EL PASO, 1775 BROADWAY, N. Y. C.

AND ITS SUBSIDIARY COMPANY

COLORADO IRON WORKS CO.

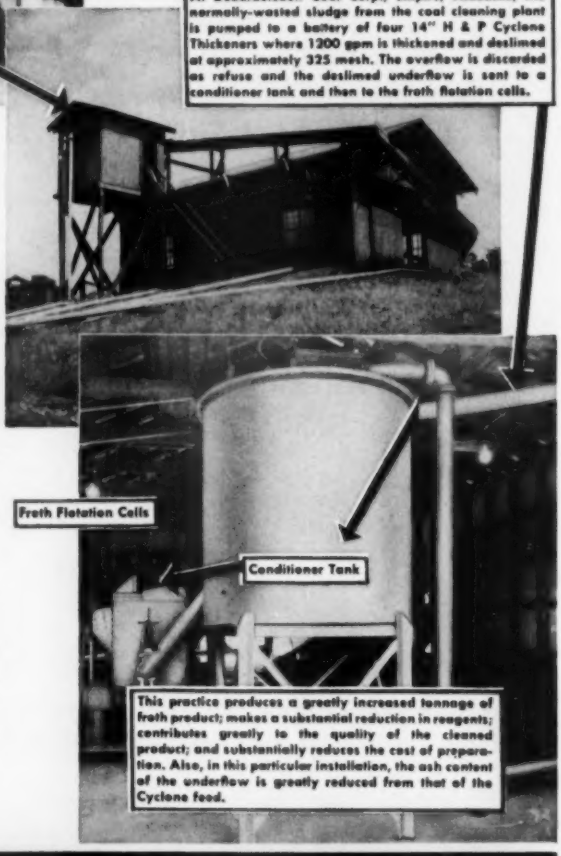
Denver 2, Colorado

Sales Agents and Licensed Manufacturers in Foreign Countries

DESLIMING Ahead of FROTH FLOTATION with H+P CYCLONE THICKENERS

Increases Froth Product and Reduces Preparation Costs

At DeBardeleben Coal Corp., Empire, Alabama, the normally-wasted sludge from the coal cleaning plant is pumped to a battery of four 14" H & P Cyclone Thickeners where 1200 gpm is thickened and deslimed at approximately 325 mesh. The overflow is discarded as refuse and the deslimed underflow is sent to a conditioner tank and then to the froth flotation cells.



This practice produces a greatly increased tonnage of froth product; makes a substantial reduction in reagents; contributes greatly to the quality of the cleaned product; and substantially reduces the cost of preparation. Also, in this particular installation, the ash content of the underflow is greatly reduced from that of the Cyclone feed.

The use of Heyl & Patterson Cyclone Thickeners by the DeBardeleben Coal Corporation at Empire, Alabama for desliming ahead of froth flotation marks another successful application of this versatile equipment in the mining and preparation fields.

H & P Cyclone Thickener equipment is a proven means of controlling solids in circulating systems and separating solids from liquids. H & P Cyclones also provide an economical and practical answer to Classification and Thickening problems.

Heyl & Patterson laboratory and field tests have provided a sound basis for predicting accurately the Cyclone performance on most feeds. For more information send for Booklet 5-CT-51 or send us your problem and we will tell you how an H & P Cyclone application can provide the answer.

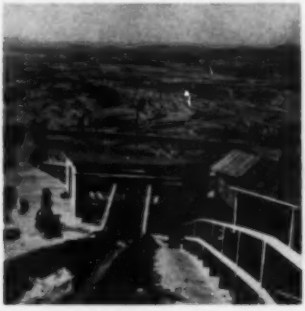
Cyclone Thickeners
Thermal Dryers
{The Drying Dutchman / Rotoverald Centrifugal Dryer}
Thorsten Coal Handling Systems
Rotary Mine Car Dumpers
Coal Crushers
Coal Preparation Plants
Bradford Breakers

Heyl+Patterson, Inc.
"SINCE 1887"
55 WATER STREET · PITTSBURGH 22, PA.

Inco Places New Saw Mill in Operation

The new saw mill at Cache Bay of the Geo. Gordon & Co., Ltd., a subsidiary of International Nickel Co., of Canada, is equipped with the first fully electric saw carriage drives in Ontario.

It has a capacity of 120,000 ft of lumber in an 8-hr shift. About half of the mill's production is high grade white pine lumber which is shipped to the general market. The other half, red pine is used in the mining operations of International Nickel in the Sudbury district, 55 miles west of Cache Bay.



The log haul-up chain is conveying a red pine log to the sawing floor of the new Geo. Gordon & Co. saw mill at Cache Bay, Ontario. Eventually, the log will be converted into timber for the mines of International Nickel Co., parent organization of Gordon & Co.

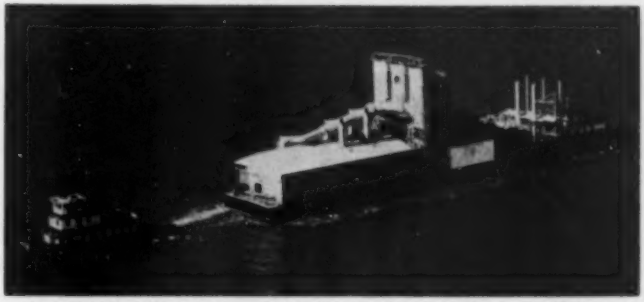
Inco with a potential need of 77 million board ft a year, gets only part of its supply from the new mill. The milling firm became an Inco subsidiary Jan. 1, 1948.

Deep Ruth in Full Production by 1953

Kennecott Copper Co. expects to have its Deep Ruth mine, Ruth, Nev., in full production by 1953, according to J. C. Kinnear, Jr., general manager.

Shafts are being deepened to reach a copper orebody lying almost 1000 ft below similar deposits at the nearby Ruth copper pit. The deposit has been estimated at nearly 23 million tons. The main Ruth shaft is being sunk to a depth of 1700 ft.

The adjacent Kellinske shaft will go down to nearly 2000 ft. Ore will be removed on three levels of the main shaft, with a daily production expected to be about 8000 tons.



Freeport Sulphur Co. will mine a sulphur deposit at Bay Ste. Elaine in the Louisiana Marshland with this plant mounted atop a 200-ft steel barge. The barge cost more than $2 million. Freeport will use sea water in Frash process extraction of the sulphur. Production is scheduled to begin soon.

Falconbridge Nickel Ltd. Gets $5 Million Credit

The Export-Import Bank established a credit of $5 million for the Falconbridge Nickel Mines, Ltd., for expansion of nickel, copper and cobalt production.

Approximately half of the total cost of the expansion program will be financed by Falconbridge, with the remainder of the funds coming from the Defense Materials Procurement Agency and the Export-Import Bank. Extent of DMPA financing is unknown.

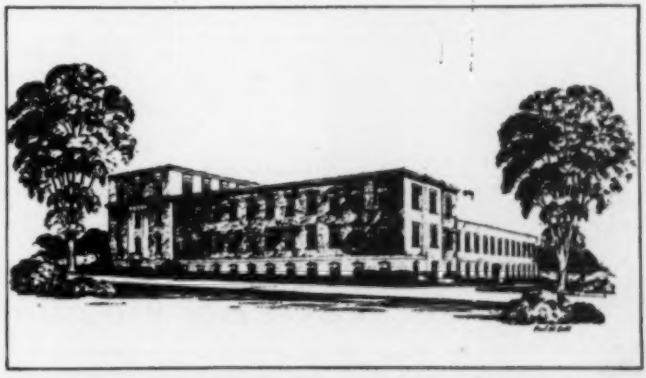
The new credits will be used for deepening the present main shaft and sinking a new one at the company's Falconbridge mine in the Sudbury district of Canada. The financing will also be used to develop and equip the firm's new Hardy mine, and to expand the capacity of its mill and smelter to produce a minimum of 35 million lb or more of combined nickel and cobalt annually.

Falconbridge is under contract to DMPA for delivery of not less than 50 million lb of nickel and 1.5 million lb of cobalt by 1961. It also has an option to deliver an additional 25 million lb of nickel and copper.

San Manuel Gets RFC, Magma Copper Loans

Magma Copper Co. will furnish its subsidiary, San Manuel Copper Co., with $8 million, in addition to a $94 million Reconstruction Finance Corp. loan, for development of the Arizona low grade copper deposit.

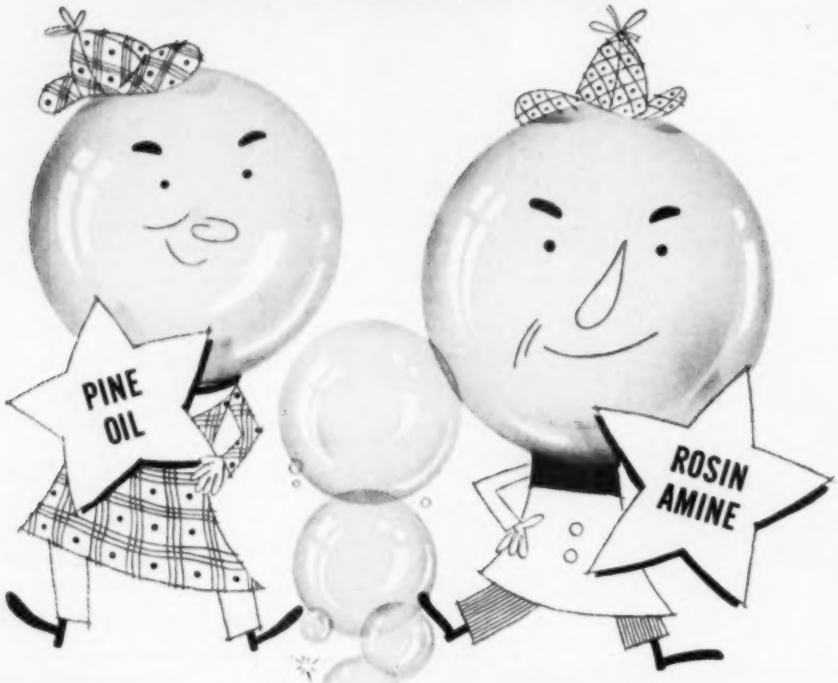
Magma has already invested an estimated $11,342,000 in the San Manuel property. Development of the San Manuel mine indicates tonnages of higher grade ore expected to lengthen the life of the mine at a lower cost of operation.



The General State Authority is currently completing the Mineral Science Building at Pennsylvania State College. The building will furnish space for research in the fields of beneficiation of coals, clays, iron ore, glass sand and submarginal minerals.

Use these "Agents"

FOR BEST RECOVERIES



Frothers

HERCULES YARMOR® F PINE OIL
Standard of quality among pine oil frothers since the early days of flotation. Yarmor F Pine Oil produces a strong froth of good volume, satisfactory texture, and excellent cell-life stability.

Collectors

RADA (Hercules Rosin Amine D Acetate)
Siliceous mineral collector that has good solubility and handles easily.

Rosin Amine Derivatives: Special purpose collectors; uses vary with type of materials to be floated.

Technical information on these and other flotation aids is available. Send for 16-page booklet "Flotation and Hercules Flotation Agents."

HERCULES Flotation Agents

HERCULES POWDER COMPANY Naval Stores Department, 955 King Street, Wilmington 99, Delaware

NM52-1

"Operation Headframe" Saves Weeks at Friedensville

Headframe Moved 67 Ft To Shaft After Erection

Towering 134 ft into the sky, 200 tons of steel headframe inched forward from its construction site, 67 ft from the shaft of New Jersey Zinc Co.'s new Friedensville mine. Mounted on mine flat cars and highly polished, greased steel rails, the huge mass of steel traveled slowly to its destination. Almost without vibration, two truck winches drew it steadily forward. Insuring complete control, a crane winch was connected to the back stay section.

Three hr and 20 min net time after the operation began, Austin Co. engineers and builders saw the culmination of months of planning and construction. The shaft was accessible a few hours after the move was completed, being out of service for only two days during the removal of the temporary headframe and installation of the permanent one. Moving the permanent headframe into place, fully erected, saved four to five weeks, based on normal working hours.

Originally, the plan was to build the permanent headframe around the temporary structure. Schedule changes forced abandonment of the idea to avoid interference with mining operations. As shaft sinking progressed, plans for moving the permanent headframe from construction site to shaft crystallized.

A temporary foundation was built 30 ft from the mine shaft. Add the 37-ft width of the headframe and it makes a distance of 67 ft over which the structure had to be hauled. Four 90-lb rails were placed on each of the two foundation lines under the main structure columns. They held the main weight of the headframe during construction and when extended on the west end, provided skidding surface for movement.

The back-stay section of the headframe was constructed with the inclined members supported at the bottom by wooden cribbing and with the horizontal members resting on three coupled flat cars. The flat cars carried the comparatively light load of the back-stay section during the moving operations, traveling on industrial gage track and 30-lb rail, with ties on 18-in. centers. The headframe was stiffened with steel struts welded to the 10-in., 49-lb columns, and tied diagonally with cable steadied by guy wires.

The headframe and permanent foundation were not originally designed for the kind of operation that took place. Thus, allowances had to be made in moving for placing the structure accurately. The headframe was moved at sufficient height to clear the anchor bolts.

Jacks, some of which were placed on the foundation, while others rested on the adjacent ground, were used to lower the headframe into place. Concrete footings were formed next to the foundation to insure equal settlement. Jacking beams were added to permit operation of four jacks around each column.

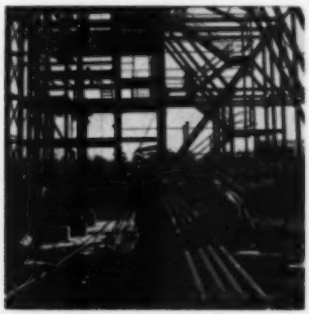
Moving speed was 6 in. per min at the outset. While friction coefficients depend upon the amount, type, and temperature of the grease used, preparations were made to pull the headframe with a 76,000-lb force, with 0.2 as the friction coefficient. It was estimated that two thirds, or 51,000 lb of the applied force would be needed at the center line of columns, necessitating an imbalance between the two sheave-systems used.

Main-lift, pillow-block supports were placed on the headframe and ground turning sheaves were installed before the move began to reduce shaft outage after movement completion.

Headframe and accessory equipment were designed for a total lift of 100,000 lb, including cars, cables, and live loads, from the bottom of the 1260-ft shaft. E. J. Longyear Co. sank the shaft for New Jersey Zinc.

The temporary wooden headframe on the left was removed two days before the moving operation began. The permanent steel headframe was moved 67 ft in 3 hr, 20 min, net time, and the shaft was tied up for only a few days.

The comparatively light back stay section was mounted on mine flat cars during construction. Industrial gage track, and 30-lb rail with ties on 18 in. centers were used. Struts, cables and guy wires were used to stiffen the headframe.



The skid rails were greased before the movement of the headframe started. A complete luff-rigged tackle lies to the left of the rails. Moving speed was 6 in. per min at the start. The friction coefficient was estimated at 0.2.

The high mechanical advantages of the rigging caused the luff blocks to come together several times. Riggers shift the cable clamps in preparation for the next move. Foot marks on the skid rails aided in moving the headframe evenly.

Shaft Operations Start Again Soon After Move

Two Special Unloaders Built by Dravo Corp.

Two specially designed unloading structures are being built for handling raw materials for the new Fairless Works near Morrisville, Pa., at an unloading slip on the Delaware River.

The unloaders are identical and can be used for either iron ore or coal. The unloaders move on rails alongside the ship. Dravo Corp.'s engineering works div. is building the unloading structures. U. S. Steel Co.'s American Bridge Div. is erecting them.

The unloaders are equipped to handle three sizes of buckets; a 15-net-ton capacity unit for ore, an auxiliary 7½-ton ore bucket, and 10-ton coal bucket. Four receiving hoppers are built within the framework of the towers. Two, 101 tons capacity each, unload ore into rail cars, while the other two feed to conveyor belts, one for coal and one for ore.

The main tower supports a boom extending 109 ft over the water and carries the bucket trolley and operator's cab. The tower rides on four electrically driven eight wheel trucks, one at each corner. The trolley boom is hinged so it can be lifted to an almost vertical position to provide clearance for ship superstructures.

U. S. Steel ore ships from Latin America will start unloading in the Delaware as soon as channel depth reaches 25 ft. The direct shipments may mean the biggest year in the history of the Philadelphia port.

Facilities at the port, and those constructed by the steel company, and those under construction by the railroad will allow the port to handle an almost unlimited amount of ore.

SLY PIONEERS & LEADERS in INDUSTRIAL DUST CONTROL

These 9 Sly Dust Filters collect and reclaim all of the dust from 70,000 cu. ft. of air per minute, 3 tons per hour of copper sulphide ore in the crushing and grinding section of the oxide plant of the Chile Exploration Co.

100% DUST COLLECTION with SLY DUST FILTERS

The above is one of thousands of successful installations. Sly Dust Filters are widely used in mines, above and below ground, where dust is a problem of safety, cleanliness or the saving of a valuable product.

Sly Dust Filters collect the dust by filtration through cloth and *are guaranteed to collect all of the dust.* They are furnished complete with fans, motors, supports, conveyors, etc.

Tell us your problem and we shall be glad to make recommendations without charge.

THE W. W. SLY MANUFACTURING CO.
4783 TRAIN AVENUE • CLEVELAND 2, OHIO
BRANCHES IN:
NEW YORK • CHICAGO • PHILADELPHIA • SYRACUSE • DETROIT • BUFFALO
CINCINNATI • ST. LOUIS • MINNEAPOLIS • BIRMINGHAM • LOS ANGELES • TORONTO

MIT Changes Name Of Geology Department

The Massachusetts Institute of Technology has renamed the Department of Geology the Department of Geology and Geophysics in recognition of the increased emphasis on instruction and research in geophysics within the department.

The department offers two separate courses, one leading to the degree of bachelor of science in geology and the other to a degree of bachelor of science in geophysics. Either course satisfies requirements for admission to the MIT Graduate school for study toward advanced degrees in geology, geophysics and geochemistry.

Uranium Strike Made On Isles Indians Fear

A group of northwestern Ontario Islands which Indian tribal legend say are cursed, have produced a uranium strike which is described as being of more than passing interest.

Ontario Minister of Mines Philip T. Kelly announced the find on the Manitou Islands at the eastern end of Lake Nipissing. Samples tested after several months of Geiger counter surveys ran as high as .11 pct uranium.

Indian legend tells of braves who made war paint from a red oxide found on the islands and later died of a bone disease.

Phosphate Plant Under Construction in Idaho

Work has begun on preparation of the site of Western Phosphates Inc. phosphate plant at Garfield, Utah. The new company has capital of $5 million and is 50 pct-owned by Stauffer Chemical Co., of San Francisco.

Kennecott Copper Corp., and American Smelting & Refining Co. own 25 pct interests. The plant will produce concentrates, superphosphates, and phosphuric acid, with most of the output sold as fertilizer. Phosphate rock will come from Stauffer's Wyoming and Idaho deposits.

Coal Safety Board Denies First Appeal

The Coal Mine Safety Board of Review, created under the provisions of the Coal Mine Safety Act denied appeal of the Morrisdale Coal Mine Co., from a ruling by Federal inspectors.

The inspector classified the mine, located in Pennsylvania, as "gassy" after taking air samples from the mine, finding one of them to contain .44 pct methane. The remaining three contained less than .25 pct methane.

The company did not contest the finding but argued that the mine should be in a position to be classified as non-gassy on a later inspection. Bureau of Mines counsel held that the Act requires the director and the board to uphold the classification when a Federal inspector finds more than .25 pct methane present.

New Self-Unloading Ore Carrier Makes First Trip

Reynolds Metals Co.'s new ore carrier, which unloads directly onto dock facilities brought its first cargo of 13,000 tons to Mobile, Ala., from Reynolds Jamaican deposits.

The freighter is named for the late O. C. Schmedeman, chief geologist for Reynolds who first discovered the significance of the Jamaican bauxite deposits and recommended that his company begin mining operations. The Schmedeman can unload its cargo in 8 hr, about one-third the time for ordinary ore carriers.

The ore is moved to the ship's stern on a built-in conveyor system, carried onto a dockside belt, into a hopper, and then to railroad cars.

Eventually, the carrier will unload at the new Reynolds plant now under construction at Corpus Christi, Texas. Currently, the ore is being shipped by rail to the Reynolds alumina plant at Hurricane Creek, Ark.

Anaconda Aluminum Co. Organization Completed

Cornelius F. Kelley, chairman of the board of Anaconda Copper Mining Co., announced completion of organization of Anaconda Aluminum Co., a subsidiary.

The aluminum reduction plant will be built near Columbia Falls, at the base of Teakettle Mountain, Flathead County, Mont. It will have two potlines and accessory buildings and equipment capable of producing 50,000 tons of aluminum annually.

Preliminary construction of the $45 million project began in September. Bulldozers are working on the area and a temporary power line is being installed. The Great Northern Railway is building a spur to the plant site. The plant is scheduled for completion early in 1954. Power will come from Hungry Horse Dam through the Bonneville Power Administration.

Francis O. Case, a vice president of Anaconda Copper, has been elected president of Anaconda Aluminum.

Mexican Gulf Plant In Operation by Feb.

Mexican Gulf Sulphur Co., in a statement issued by Paul Nachtman, president, will have its plant on the Isthmus of Tehuantepec in Vera Cruz in operation by early 1953. The company has the San Cristobal sulphur dome concessions.

Nachtman announced that his firm would be the first to use the Frasch process in Mexico. More than 95 pct of the engineering has been finished and more than 85 pct of the machinery has reached Mexico. The plant will have a minimum annual capacity of 200,000 long tons, with a goal of 500,000 long tons by 1955.

In the meantime, Pan American Sulphur Co. announced plans to build a $5 million plant at Jaltipan in Southern Vera Cruz. The plan will also use the Frasch process. Construction is expected to begin in February, with completion by August 1954.

CF&I

PRODUCTS FOR THE MINING INDUSTRY:

Cal-Wic Wire Cloth Screens

Mine Rails and Accessories

Rock Bolts

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Grinding Balls

Grinding Rods

You'll find a CAL-WIC Screen for every mining and industrial purpose. They're woven to the most exacting tolerances to fit any specification, and can be obtained in ferrous and non-ferrous metals to meet individual requirements. Many types are available from stock.

For complete information on CAL-WIC Industrial Screens write today for Catalog 1249. Address the office nearest you.

THE CALIFORNIA WIRE CLOTH CORPORATION, OAKLAND
THE COLORADO FUEL AND IRON CORPORATION, DENVER
WICKWIRE SPENCER STEEL DIVISION, NEW YORK

1138

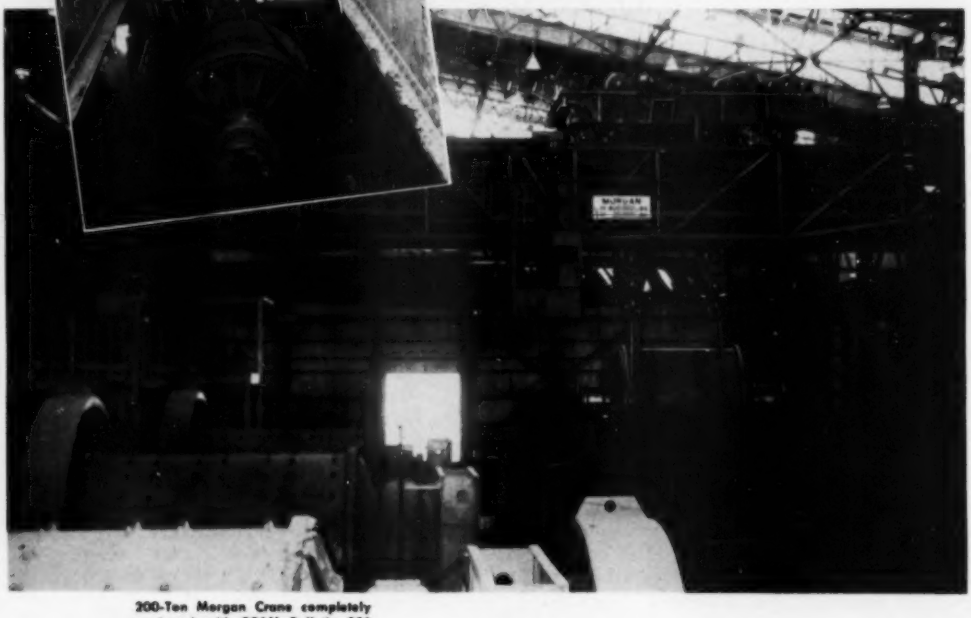
CAL-WIC
INDUSTRIAL SCREENS CF&I

CRANE CONTROL

These EC&M Engineered Features provide —

1. Greater accuracy in spotting ball or rod mills.
2. Safer handling of HOT metals.
3. Limited torque to avoid damage to cranes or converters when removing "skulls."

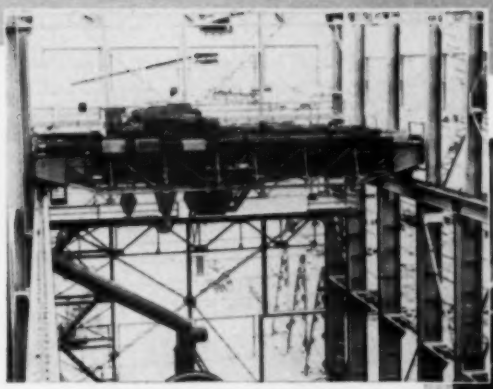
EC&M Control enables operator to spot ball mills accurately; crane also handles heavier rod mills.



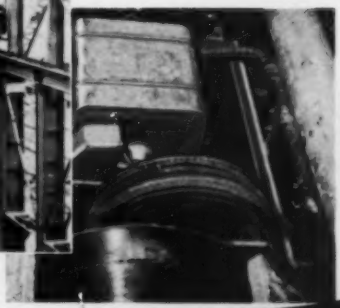
200-Ton Morgan Crane completely equipped with EC&M Bulletin 921 Magnetic Controllers, Bulletin 1004 Type WB Brakes and Bulletin 1032 Youngstown Safety Hoist Limit Stop.

THE ELECTRIC CONTROLLER & MFG. CO.

Speeds Copper Output



One of two 100-Ton Morgan Cranes installed in the converter aisle, both equipped with EC&M Control for safe, sure handling.



Top view of one of the crane trolleys showing EC&M Bulletin 1032 Youngstown Safety Hoist Limit Stop for prevention of overhoisting accidents.

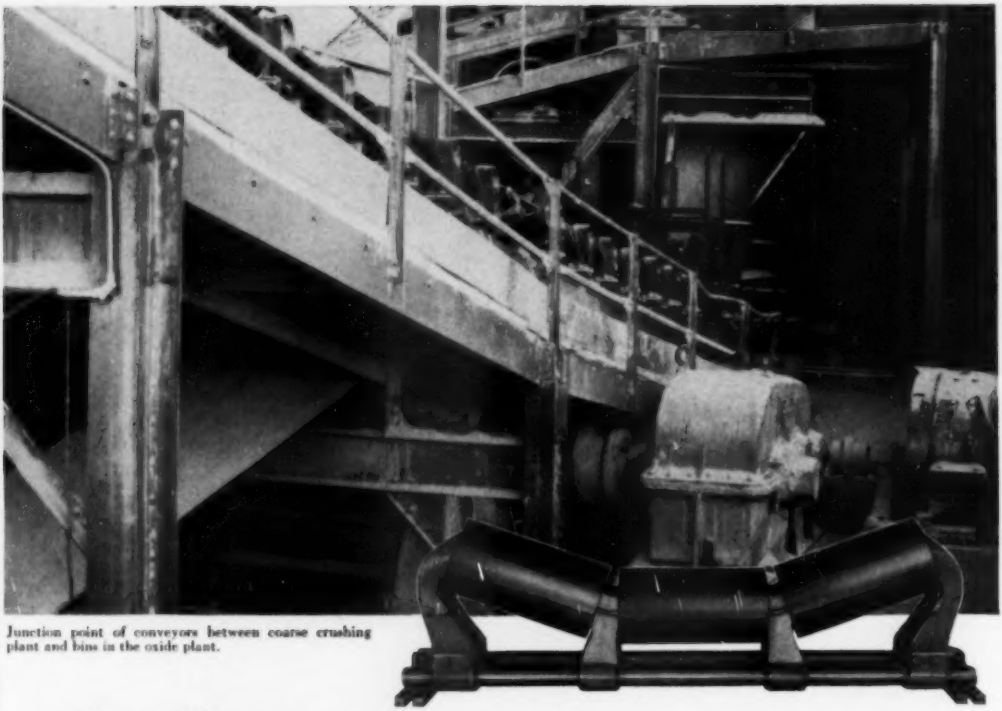
★ On Pumping Stations EC&M Motor Starters Are Oil-Protected to Prevent Corrosion—Reduce Maintenance—Are Ever Ready to Start

EC&M 550 volt, Bulletin 1045 Auto-transformer Starters in sea water pumping station at Tacoma power plant.

EC&M 2300 volt VALIMITOR Motor Starters in filter plant.

EC&M 2300 volt, Bulletin 1062 Type ZMS Motor Starters in reclaimed water pumping station.

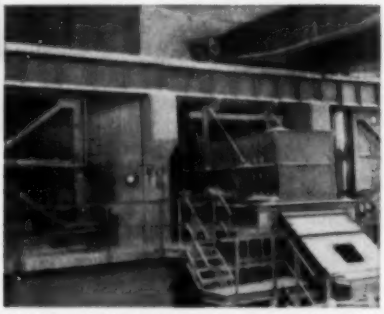
2698 East 79th Street • Cleveland 4, Ohio



Junction point of conveyors between coarse crushing plant and bins in the oxide plant.

SIX MILES of Conveyor Belts

Ride on S-A CARRIERS at CHILE EXPLORATION CO.'s Chuquicamata Plant



S-A Amsco Feeder in the secondary crushing plant. This is one of six Amsco Feeders used at Chuquicamata.

More than 2500 of these S-A Carriers have been in service for a quarter of a century. All of this group are in the multi-million ton class—*and are still going strong!*

When the huge new sulphide plant was built—and with this record to go by—enough S-A Carriers to equip 2½ miles of new conveyor belts were installed.

Most of the Carriers used are S-A "444" Heavy Duty Carriers especially designed for maximum operating economy and long life. Their unique "tip-back" feature enables Carriers to be inspected without shutting down the conveyor. Other Carriers on the job are S-A "Mammoths," probably the largest anti-friction units ever built.

Besides its Carriers, S-A has six Amsco Feeders on the job in the crushing plant.

We at S-A feel that there could be no greater tribute to the worth of S-A equipment than to be chosen—*and chosen again 25 years later*—by Chile Exploration Co. for its Chuquicamata operation.

STEPHEN S-A ADAMSON MFG. CO.

4 Ridgeway Avenue, Aurora, Illinois — Los Angeles, Calif., Belleville, Ontario

DESIGNERS AND MANUFACTURERS OF ALL TYPES OF BULK MATERIALS HANDLING EQUIPMENT

P&H Electric Shovel in the pit of Anaconda's Chile Exploration Co.

in the pits

in the plants

P&H Overhead Crane in grinding bay at concentrator of Anaconda's Chile plant.

of Chile Exploration Company

P&H IS SPEEDING COPPER ON ITS WAY!

Copper's on the increase . . . coming faster than ever from the Chile Exploration Company. And, P&H is there — helping all along the way!

In the open pits, fast-digging P&H Electric Shovels are feeding huge tonnages of ore to the plant . . . and doing it at rock-bottom costs! For here, as in mining operations everywhere, the greater speed, strength and stability of P&H Electric Shovels mean lowered maintenance and greater output. More advanced design accounts for it — with features like famous P&H Magnetorque* Hoist Drive which *adds* speed and smoothness to hoisting motions, *eliminates* all mechanical devices.

In the plants, P&H Overhead Cranes provide swift, sure handling of materials. Serving in the grinding, regrind and flotation bays, P&H Cranes — job-engineered by America's leading crane craftsmen — perform better and for less with all the dependability that comes from such exclusive advantages as P&H-built electrical equipment.

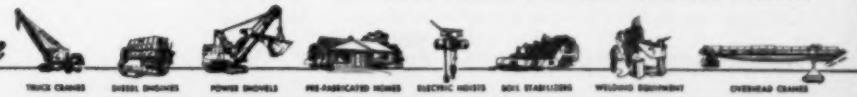
Ask for full information on P&H Electric Shovels and Overhead Cranes — both leaders in their fields!

*T.M. of Harnischfeger Corporation for electro-magnetic type coupling

HARNISCHFEGER CORPORATION

4400 WEST NATIONAL AVENUE, MILWAUKEE 46, WISCONSIN

the [redacted] *Line*



Sulphur Decontrolled——Zinc Surplus——New Uses for Old Mines——Steel Denationalization

THE National Production Authority removed limitations on use of sulphur and eliminated inventory controls. According to officials supply had caught up with demand. The improved supply condition came about largely through a decline in actual consumption which fell below authorized use during the first eight months of 1952. The limitation order restricted sulphur use to 90 pct of 1950 usage and provided for certain defense and essential civilian requirements. Estimated sulphur consumption in all forms, including exports in all forms with the exception of spent sulphuric acid is 6,090,000 long tons, compared with an estimated supply of 6,524,000 long tons.

But the control situation is not as bright in other branches of the minerals and metals field. The steel and aluminum industries are almost unanimous in their desire to see the end of controls. While there is still no overbalance in the supply of either metal, the tight supply situation has been steadily easing in various products and new production capacity is coming into being. The steel industry has installed new annual capacity of about 6 million tons. The aluminum industry has been keeping pace with a capacity of approximately 550 million lb.

Some industry experts feel that if controls are discarded and red tape torn away, much of the current demand will prove to be more mythical than actual. In the case of aluminum foil the supply picture showed few signs of easing until NPA dropped its restrictions. Within a week, supply exceeded demand. The aluminum industry seemed to have been set for an offensive against controls until the northwest power shortage set in. Now, it appears that they will make their onslaught in December.

Military demands on steel have been about 10 to 12 pct of production and about 50 to 55 pct of aluminum output. With or without restrictions, it is said, these needs are given first priority.

Copper producers, on the other hand, are more concerned with the lifting of price ceilings than with controls. They consider price ceilings discriminatory. Some people believe that price controls will go out the window this year, allowing domestic price to seek its own level. Some Government officials are said to be in agreement with the idea. One official believes that after a short period of confusion price will level off at about 30¢ and custom smelters will go to about 36.50¢. However, before the Government can present a program of action, agreement must be reached between OPS, NPA, DMPA, GSA, and the State Department.

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JESS LARSON, Defense Materials Procurement Administration declares that the surplus in zinc which appears to have developed is in truth artificial. He states that unless production of lead and zinc can be maintained serious shortages are certain to develop next year. Larson said, "the freeing of the London lead market and reports that the British Government contemplates removing its controls over the zinc market about the first of the year have had the effect of creating an artificial surplus of the two metals."

Several marginal and small zinc mines have been closed down recently, but market sources claim that it will be six months before the effect on production will be known. But as the market continued weak, there was a strong possibility of further mine closings.

Larson has been questioned on the Government's intentions. "On the basis of all available evidence, the present price situation is of a temporary nature. I am sure that actual demand will catch up with artificial surplus in the near future."

The Defense Production Administration, in the meantime, has cut 85,000 tons from its expansion goal for zinc capacity. The new goal calls for a total supply of 1,245,000 short tons in 1956.

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WHAT happens to an iron mine when it no longer can produce ore at a profit? Usually, it just goes to seed. The shaft is likely to cave after a time, the timber rots, and the mine fills with water. Its economic use becomes a thing of the past. But not so with an abandoned mine near Livingston, N. Y.

The Iron Mountain mine produces nothing today but security. Its intangible product is a child of the atomic age. It has become a storehouse for blue chip records, valuable art objects, and other things that people fear will be irreparably lost in the advent of an atomic war.

Owned by Herman Knaust, it has been named the Iron Mountain Atomic Storage Corp., with 117 reinforced concrete vaults. Iron Mountain is air conditioned and has its own standby electrical system.

It brings to mind an abandoned copper mine in Michigan's Keweenaw Peninsula Copper Range country. The mine was taken over by a resort operator and has a good solid adit level. The resort man claims he makes more money from showing the underground workings during the summer than the mine was able to make in the days when it produced copper ore.

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BOLIVIA, after seizing the mines of its three largest tin producers, is trying to convince the world that it is not antagonistic toward foreign investors. Victor Andrade, Bolivian ambassador to the U. S. said that the Patino, Hochschild, and Aramayo tin properties nationalized by his Government were a special case and that the action had no bearing on other foreign investors in the country.

The Ambassador said the nationalization was meant to "eliminate forces that have made it impossible to have a democratic and stable government in Bolivia." He laid the responsibility for poverty of the workers at the feet of the tin companies. In most quarters his statement was viewed as a rather

broad picture of the situation, which required a great deal of filling in.

Small, privately owned tin mines have not been affected by the decree. Railroad properties controlled by British capital have not been affected. The ambassador pointed to a projected tin smelter to be built with Argentine money as an indication of Bolivia's welcoming of foreign investment.

In the meantime, American officials have serious doubts that Bolivia will be able to operate the nationalized mines. It is feared that a serious blow will be administered to tin concentrate delivery to the U. S. An unofficial estimate has already reduced expected deliveries for 1952 from 15,000 tons to 10,000 tons.

Bolivia supplies only about 20 pct of the world tin, but she has become vitally important in American defense plans. In order to mine at a profit, a combination of low and high grade ores must be extracted. It is feared that the Government will have a tendency to "gut" the mines, leaving the low grade ore in order to gain a large profit fast. It is questionable whether the men with the know-how, mostly Canadian and American engineers, will remain in Bolivia.

At the Hague, the International Tin Study Group announced that Bolivian tin exports in August fell 160 tons below the 2969 tons shipped in July. At the moment, nothing is certain, not even the amount the three companies will receive as compensation. Bolivia has promised to pay, but at the same time claims the companies owe her $520 million. The decree nationalizing the mines set forth provisional sums of about $21.75 million to be paid the companies as indemnities.

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MINNEAPOLIS-HONEYWELL Regulator Co., and Trans-Weigh Co., have come up with a conveyor belt that thinks. The belt weighs material and then figures out the tons-per-hour that it carries. And you can't fool it by adding or subtracting material along the way. It makes allowances for any additional material as well as for any loss.

The belt rides along a set of rollers. The force exerted on the rollers is transferred to a gage. The weight of the belt is subtracted electrically. Final measurements are fed into an electronic recorder which indicates the flow at the moment and the total amount which has been carried by the belt.

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BRITAIN'S Tory party is moving fast in its desire to return the steel industry to private ownership; almost on top of a measure to denationalize the trucking industry. Winston Churchill's Conservative Government is committed to a policy of undoing the work of the Labor Government and in no uncertain terms it is setting about the task of accomplishing the steel denationalization job. The Tory bill is no surprise. Prime Minister Churchill's conservative party has already published a white paper outlining the general program.

It has been recognized that the business of getting the Government out of steel is a monumental task which will take some years to accomplish. Tories said that in deciding the asking price for nationalized properties, the Government would take into consideration changes in trading prospects, reserves and other matters since nationalization took place. Denationalization does not promise complete freedom from government. An Iron and Steel board will be set up which will have the power to exercise some control over the industry.

One definite danger seems to be present in the move to return steel to private ownership. The Labor Party has promised to undo anything the Government does in that direction if and when it is returned to office. Members of Parliament hope to put the bill into law by Easter.

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NEW MEXICO'S mining industry established an all-time production high during the fiscal year ended June 30, 1952. Output value was almost $10 million more than in the previous fiscal year, with a total value for all minerals produced of $99,648,584. Potash production led the list, with 7,017,814 tons, valued at $36,117,201. Duval Sulphur and Potash Co., made a substantial investment in new facilities and is given at least part of the credit for the increase in state production. The Southwest Potash Co. is expected to start producing by October, 1953 making the predicted 10 million tons of potash a strong possibility during the 1952-53 period.

Copper was in second place with 73,974 refined tons valued at $35,767,417. Metallic production bettered nonmetallic output other than coal. New Mexico's coal production increased 41,879 tons to a total of 844,095 tons, but definite signs of a decline in the industry can be seen. Defiance Coal Co., of Menmore ceased production after 35 years.

New Mexico also made substantial contributions to the production of vital strategic materials, including uranium, beryl, and manganese. A definite increase in uranium production is foreseen when Anaconda Copper Co.'s new mill near Bluewater is completed in June, 1953. The mill is expected to process large quantities of low grade ore from the Grants district.

More than $11 million were poured into improvements by state mine operators for underground facilities, surface construction, and equipment. Duval spent a large share of the total.

The safety record was also significant. Only five fatal accidents were reported, making one for each 3,100,685 man-hours of exposure. However, there was an increase of 168 non-fatal accidents to 638. Time lost because of accidents was lower than in the previous year.

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automatic feed drifters

—Low vibration simplifies set-up.
—Self-adjusting feed maintains high drilling speed.
—Powerful rotation handles tight steel, saves lost holes.
—Convenient controls boost drill runner's efficiency.
—Long feed guide shells for fewer steel changes.

DRIVE HEADINGS FASTER with this

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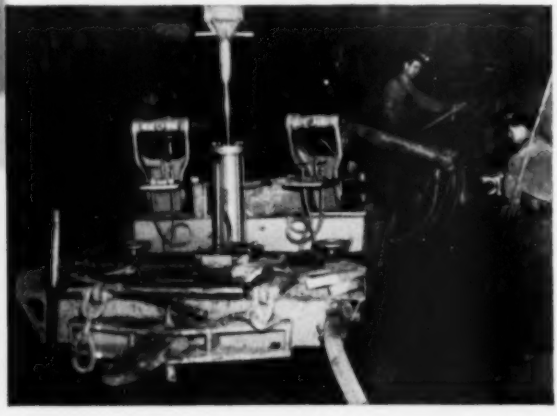
hydraulic drill jumbos

—Faster set-up—move in and start drilling.
—Long feeds—up to 10 feet—give more actual drilling time per shift.
—"Creep-free" booms — maintain steel alignment.
—Offset booms spot lifters easily.
—Faster tearing down — release hydraulic pressure and back away.
Write today for full details.



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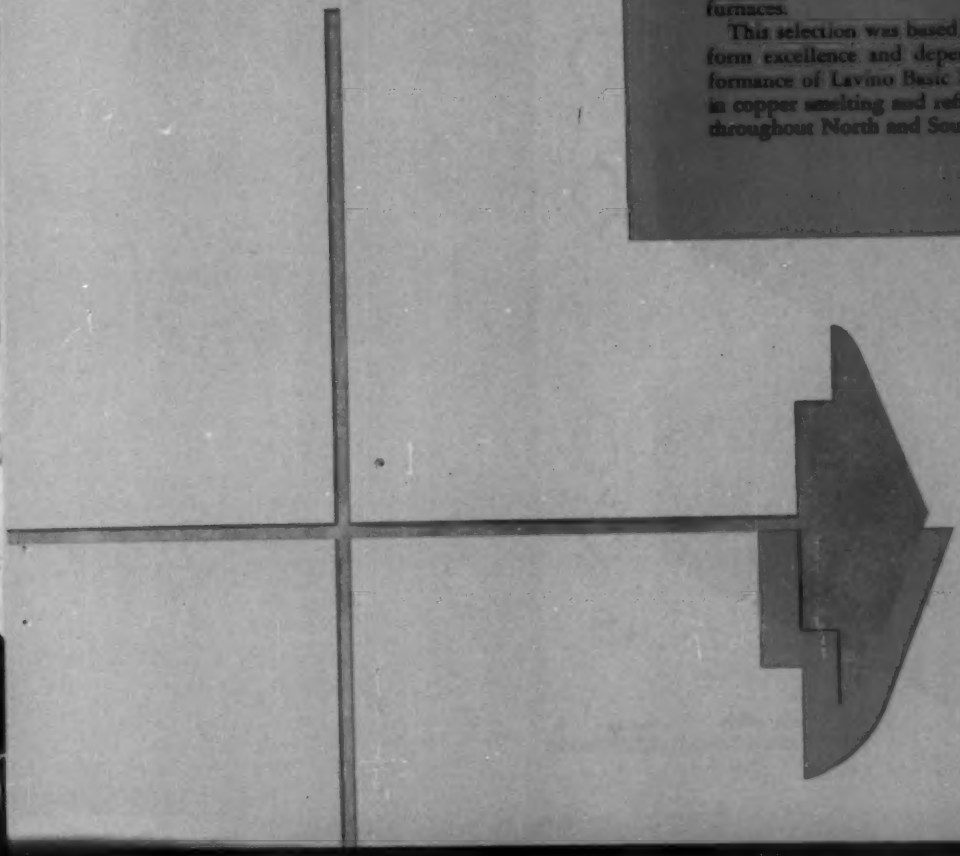
THE QUALITY LEADER IN COMPRESSORS, PUMPS AND ROCK DRILLS

"LAVINO"

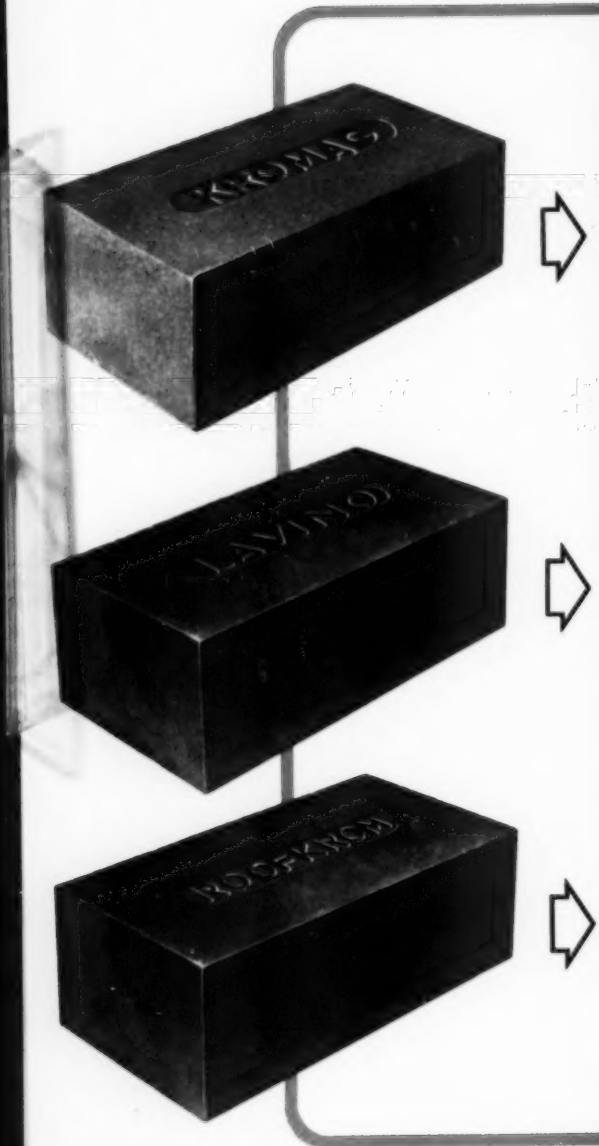
BASIC REFRACTORIES
for the
COPPER INDUSTRY

For the new Chuquicamata Plant Chile Exploration Company selected "Kromag" Brick and other Lavino Basic Refractories for lining converters, holding furnaces and certain applications in the reverberatory furnaces.

This selection was based on the uniform excellence and dependable performance of Lavino Basic Refractories in copper smelting and refining plants throughout North and South America.



"LAVINO" BASIC REFRACTORIES..



"Kromag"® is the burned chrome-magnesite brick that has revolutionized refractory practice in the copper industry. "Kromag" Brick performance is superior to other basic brick because of their higher refractoriness, lesser spalling tendency, lower thermal conductivity, greater hot load strength and resistance to hydration. Their chemical and physical stability in the presence of highly siliceous as well as basic slags enables "Kromag" to give economical service in the roofs, side walls, slag line and bottom of reverberatories, converters, fire refining furnaces (anode, cathode and wirebar), scrap copper reverberatories, electric furnaces and holding furnaces of the reverberatory or cylindrical type. "Kromag's" resistance to siliceous attack has made it possible for some converter operators to gain the metallurgical advantages of a fluid siliceous slag without seriously endangering the service life of the refractory lining.

"Lavino"® Chrome Brick is the most nearly neutral of all basic brick in chemical reaction. They are successfully used for lining settlers or forehearths, paving flues and constructing various adjacent parts of furnaces where molten slag or dust accumulates and cleaning is difficult when clay or alumina brick are used. The neutral character and high refractoriness of "Lavino" Chrome Brick resists the attack of acid or basic slag, fume and dust.

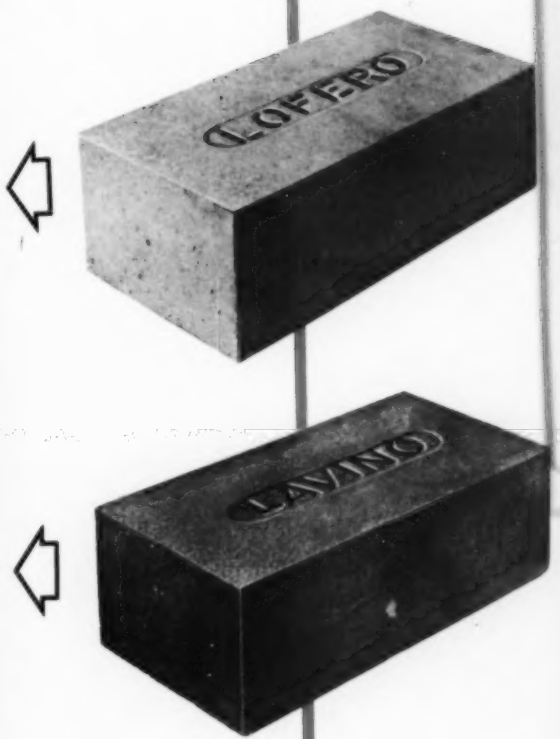
"Roofkrom"® is a burned chrome-magnesite brick especially developed for suspended and sprung arch construction in basic metallurgical furnaces. It is recommended for roof and flue arches in copper reverberatory and refining furnaces. "Roofkrom" is presently used with success for the suspended arch covering the flue leading from a wirebar furnace to the waste heat boiler and the main suspended roof of a copper scrap metal reverberatory. The choice between "Roofkrom" and "Kromag" brick for sprung or suspended roofs on copper refining furnaces will be influenced by the metallurgical practice as well as the type and size of furnace.

..FOR THE COPPER INDUSTRY

BRICK-CEMENT-GROUND PRODUCTS OF CHROME AND MAGNESITE

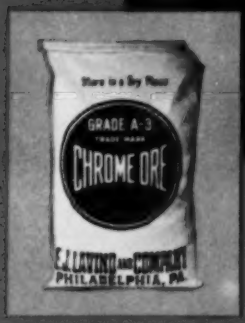
"Lofero"* is a burned Seawater periclase brick exceptionally low in iron oxide and lime and containing a minimum of 90% MgO. Its use is especially recommended for lining Dore furnaces, electric furnaces producing copper, nickel or other metals in which chrome or iron contamination is objectionable. "Lofero" brick are also ideal for lining the crucible and forehearth of copper blast furnaces.

"Lavino"* Magnesite is a burned magnesite brick produced from high purity Seawater periclase with a controlled addition of iron oxide which satisfies to some degree the magnesite's affinity for iron oxide. This product is used extensively in copper plants where operators prefer a burned chrome-free magnesite brick for lining certain furnaces.



*Registered U. S. Patent Office

"LAVINO" SPECIALTIES



Grade "A-5" Chrome Ore is a finely ground product of exceptionally high grade chrome ore. It was originally developed in 1921 and is still used as a hot patching and maintenance material. However, Grade "A-3" Chrome Ore is now also used extensively as a heat-setting mortar for laying all types of basic brick. The minimum amount of supplemental ingredients contained in this material preserves the high refractoriness of the natural chrome ore.

"Lofero"* (Periclase) Cement is a heat-setting bonding mortar for laying "Lavino" Magnesite, "Lofero" (Periclase) and "Kromag" (Chrome-Magnesite) Brick. This cement provides automatic expansion allowance through its initial drying shrinkage and slight compressibility under pressure of the expanding brick as the temperature increases. The thickness of the "Lofero" Cement joint is varied to accommodate the expansion characteristics of the different types of brick. Basic brick structures exceeding forty (40) feet in length have been installed with "Lofero" Cement without a single open expansion joint. None of the many installations of this type has shown the slightest bulging, buckling or excessive thrust on the furnace steel work.

"Hotkrome" is a granular chrome ore mix especially designed for maintaining basic, silica or clay brick furnace wall, sheets and roofs. Gunning or air emplacement is the easiest and most rapid method of application, but it is also successfully applied by shovel or spoon. Its primary advantage is its ability to stick to hot furnace walls during application and subsequent resistance to erosion. We recommend its use for maintaining slow lines and other points of severe erosion in ferrous and non-ferrous metallurgical furnaces. The relatively large grain size of "Hotkrome" makes it unsuitable for use as a mortar for laying brick.

"Kromepatch"* the original chrome bonding mortar is sufficiently neutral chemically to be safely used for laying all types of fire clay brick. It is particularly recommended for laying basic brick made of 100% chrome ore ("Lavino" Chrome Brick) and the various types of chrome-magnesite brick such as burned "Kromag" and "U.B.K."* (Unburned Kromag). "Kromepatch" is also a most effective bonding ingredient for rammed-in-place or patching mixes prepared from granular chrome ore, periclase or crushed brick bats reclaimed from furnace linings.

Plastic "K-N"* Chrome Ore is a highly refractory, chemically neutral, ready-to-use ramming material used for making burner blocks, tap hole blocks and for patching brickwork in the hearth and slag line of many types of melting and refining furnaces. Plastic "K-N" Chrome Ore is used throughout the steel and other metal working industries for rammed monolithic hearths in heating and forging furnaces, for lining water cooled furnace doors, dampers and other equipment. PK-N is the most durable refractory known for the walls and bottoms of studded water tube boilers.

*Registered U. S. Patent Office

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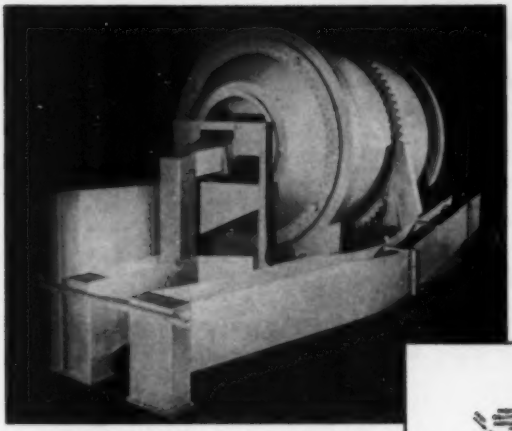
HIGHER CONCENTRATE GRADES

...lower tailings



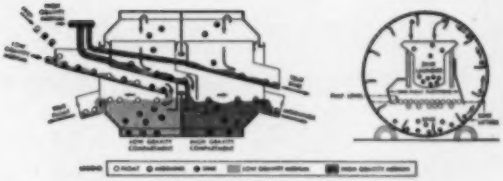
THE WEMCO DOUBLE DRUM

*is the only Heavy-Media Separator producing **three** products by **absolute*** gravity control*

Ores amenable to HMS and containing middlings are now successfully treated to produce an **optimum grade concentrate, lower tailings,** and an **accurately segregated middling** that can be reprocessed for maximum metallurgical extraction.

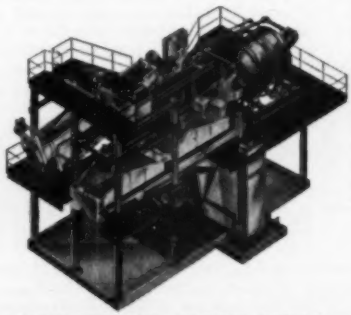
LOW CAPITAL INVESTMENT AND OPERATING COSTS

The superior metallurgical results of a two stage process are accomplished in **one** drum and **one** media reclamation circuit. There is no unnecessary duplication. Cost is a minimum.



A POSITIVE OPERATING PRINCIPLE

- True float is separated by a low gravity media in 1st compartment, while sink passes to 2nd compartment.
- True sink and middlings are separated by a high gravity media in 2nd compartment.
- Both high and low gravity media remain uniform and constant in density in their respective drum compartments.
- Overflow of float material and lifting of sink are simple, effective actions.
- The entire separation is accomplished within one unit.



AVAILABLE IN MOBIL-MILLS

For ores having middlings, Wemco Double Drum Separators are furnished as the separatory vessels with Wemco Mobil-Mills—the prefabricated HMS plants used by the majority of Heavy-Media operators throughout the world.

Ores requiring less complex treatment are beneficiated in Mobil-Mills using popular single compartment Wemco Drum Separators or the efficient Wemco Cone Separator, depending on the nature of the separation involved.

★ Absolute gravity control provides **positive** separation of the middling by the use of a homogeneous medium. This prevents the lower grades and recoveries common with the partly heterogeneous actions of other types of separating units.

Write today for information on your problem. Mobil-Mills are available in various sizes to treat up to 350 t.p.h.

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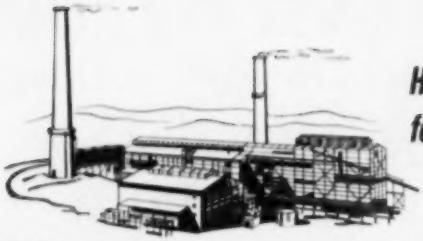

Here's a point to the Chuquicamata Story for Mine Operators and Railroad Men . . .

We've got what it takes to get your job done right!

UNISTRUT supporting power distribution lines

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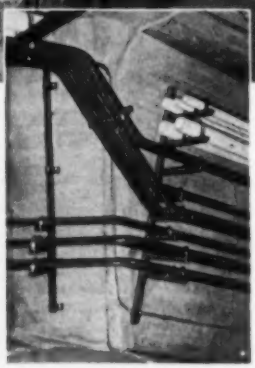
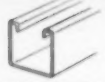
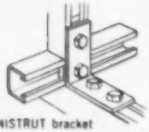




UNISTRUT channel

UNISTRUT spring nut and bolt



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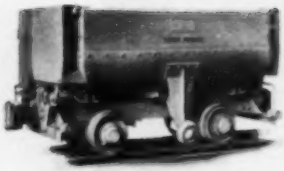

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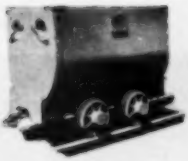

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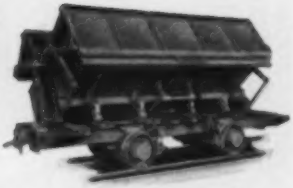

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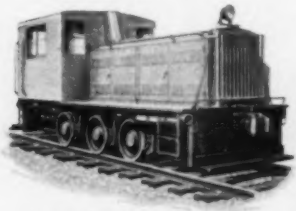

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MINING
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EDITORIAL

40-YEARS OLD, CHUQUICAMATA LOOKS TO THE FUTURE

THIS issue is about Chuquicamata and the new sulphide plant. Chuquicamata is moving into a new cycle of productivity; she has begun to give up the sulphide copper which lies deep-seated beneath the oxide ore zones in large but yet unknown quantity.

After forty years of production, Chuqui may be likened to a strapping youngster, with adolescence and maturity still ahead. Mining operations are being projected—not for decades—but for generations. Although there are still many years of open-pit mining ahead, planning of future underground mining has been started. The bottom of the orebody is still unknown, but enough work has been done to show that Chuqui is one of the largest known copper orebodies in the world.

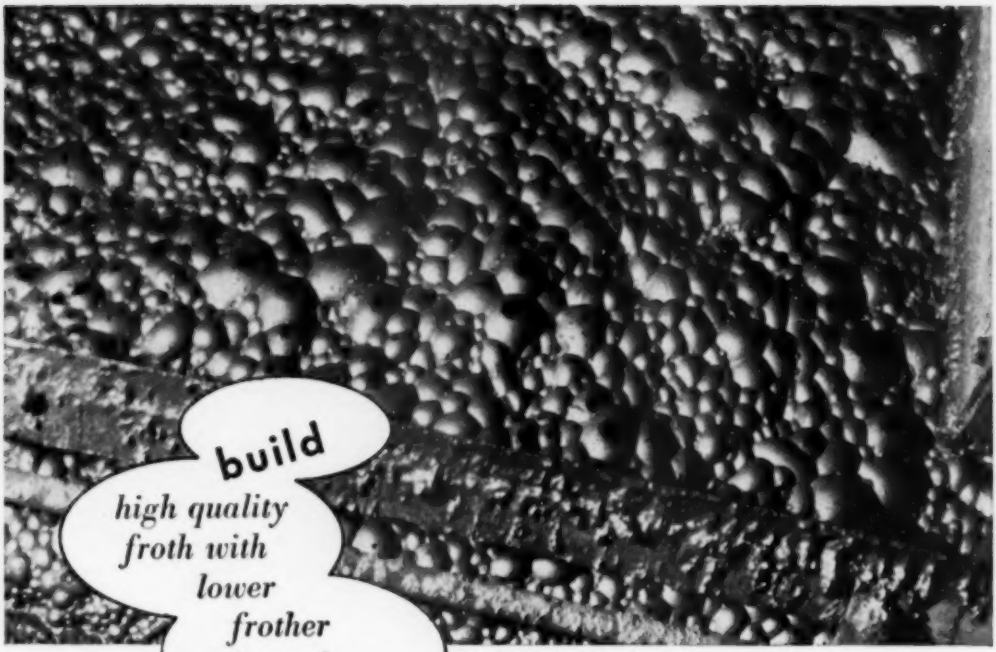
Until recently, only the oxide ores have been mined and these have been treated at the rate of 50,000 to 60,000 tons per day by leaching. Construction of the sulphide plant was timed to coincide with reduction in production of oxide ores. The new sulphide concentrator and smelter, designed in keeping with the magnitude of the orebody, will be brought on the production line at the rate of 30,000 tons per day. The capacity of the plant now planned can be expanded to treat 60,000 tons of sulphide ore.

Little wonder that Chuqui is a name legendary in the remotest mining circles. But to those who have known her more intimately, and there is hardly a mining camp without a Chuqui graduate, she means much more. She is an orebody that has made engineers. Reposing at 10,000 ft on the western flank of the Andes in the arid northern reaches of Chile, water is scarce, common fuels absent, and volcanoes and earthquakes add drama and hazard. It is these natural difficulties which have created the fascination and challenge which have caused many engineers to devote their lives to unlocking this treasure house of nature.

With the beginning of large-scale mining in 1912, management faced the problem of recruiting and training a labor force, founding a community, providing the necessities of living, and procurement of mining machinery from abroad. Early managers, recognizing the vast extent and potential of the orebody, set a pattern of permanence and integration of plant and community. Fred Hellman, 1912 to 1916, and H. C. Bellinger, 1916 to 1921, began the job. In the following 20 years (1921-1941) Burr Wheeler expanded the plant to the point where it was a leading world copper producer. From the beginning, mining and processing were planned to allow for future expansion; storage yards were provided so that inventories of supplies and spare parts could be maintained commensurate with distance and exigencies of sources of supply; complete shops were established and personnel trained to operate them to minimize dependence on distant factories; power, water, transportation, and communications were provided to meet the needs of an expanding organization. On the other hand, doctors and hospitals were brought in, schools and churches built, and channels for adequate provisioning established; the rare commodity—pasteurized milk was introduced.

Tom Campbell, 1941-1945, and Mat Sample, 1945-1948, ably managed the properties during the critical period of World War II and the postwar readjustment years. With the exception of Hellman and Bellinger, these men all began in subordinate positions at Chuqui and grew to engineering maturity on the problems of mining in a wilderness in a foreign land.

Today's manager, Charles Brinckerhoff, follows the pattern laid down by his predecessors—planning for generations rather than decades. Through the years Chuqui managers have kept abreast of modern mining methods, and today the new sulphide plant is no exception; it is the most modern in the world. *Mining Engineering* is proud to present this saga of management and engineering at its best—imaginative, daring, and responsible.



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Chairman of the Board

Anaconda Officials

Here, published for the first time, is the group of Anaconda officials (New York office) concerned with the planning and operations of the company's copper properties in Chile.

Burr Wheeler
Former Executive
Vice President,
Chile Exploration Co.

Thomas A. Campbell
Executive Vice President,
Chile Exploration Co.

Douglas M. Dunbar
Asst. to Vice President,
Chile Exploration Co.

Charles M. Brinckerhoff, busy but never ruffled, is general manager, Chuquicamata property, Chile Exploration Co.

Rodolfo Michels is vice president and director at Santiago of Chile Exploration Co. and Andes Copper Mining Co.

Wm. Wraith, Jr., assistant general manager, is a metallurgist who came to Chuquicamata from Cananea, Mexico.

History of Chuquicamata Copper

geologic eras in the making

600 years of mining history

40 years of open-pit mining

plans sulphide operations for generations

D. M. Dunbar

Assistant to Executive Vice President
Chile Exploration Company

LONG before Columbus discovered America the original inhabitants toiled in the copper workings of the Andean Cordillera. Their best diggings appear to have been at Chuquicamata, site of the huge present-day open pit owned by the Chile Exploration Co., subsidiary of Anaconda Copper Mining Co. Located in Antofagasta Province of northern Chile, Chuquicamata is in the Atacama desert, high on the western slope of the Andes at 9500-ft elevation.

In the Chuquicamata area the Atacama desert merges into the Cordillera. The climate is extremely arid and precipitation, either in the form of rain or snow, is very infrequent. The temperature varies from 50° to 85°F during the day, and drops to 20° to 40° at night.

The Antofagasta and Bolivia Railroad, which connects the Chilean Port of Antofagasta with La Paz, Bolivia, passes within approximately 6 miles of the plant. A branch from the main line to Chuquicamata serves the property for outgoing copper and incoming freight. The distance to Antofagasta by rail is 163 miles.

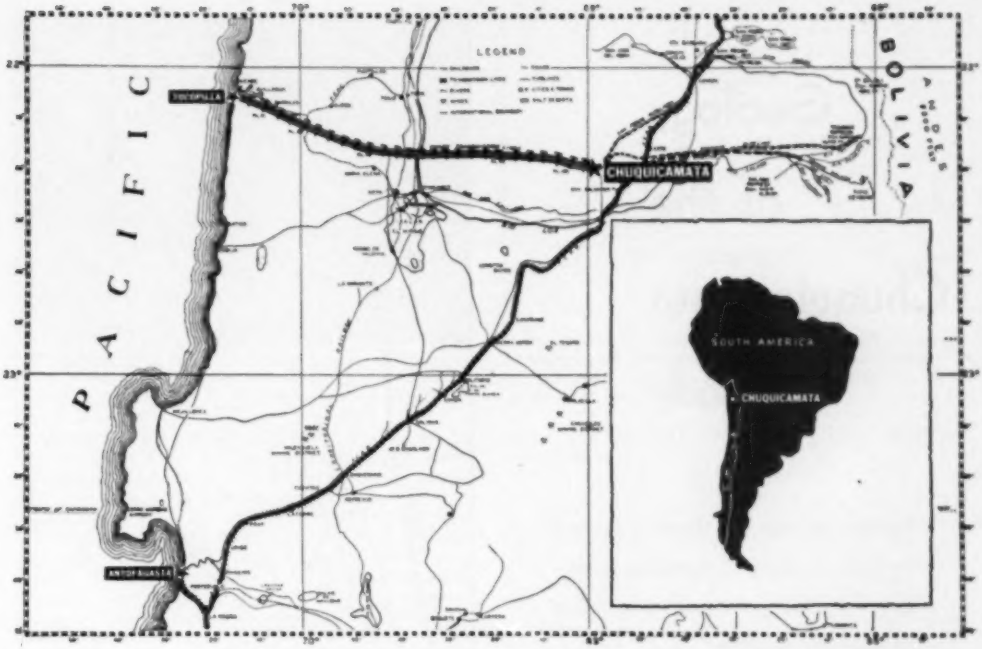
Previous to the Inca conquest, this region was inhabited by the Chuco Indians, a small tribe, descendants of the Aymaras and Quechuas.

The Incas, whose capital was located in Cuzco, Peru, gradually extended their empire to the south, reaching at one time to what is now central Chile. It is believed that they built crude primitive furnaces on the banks of the Rio Salado where on a small scale they smelted copper ore from Chuquicamata. At the time of the subsequent Spanish conquest, legend has it that Diego de Almagro, on his return to Cuzco in 1536 after his invasion of Chile, obtained from these furnaces metallic copper for horseshoes.

During the period from 1560 to 1879 the high grade veins of the Chuquicamata deposit were worked superficially by the Spaniards and Bolivians. The treaty signed after the war of 1879 recognized the sovereignty of Chile over this territory and Chile, realizing the importance of the district, started to develop it in an orderly manner, although only high grade veins were worked.

New Process Is Key

Reports on the deposit were brought to the attention of Albert C. Burrage, a Boston lawyer and financier, in 1910. Chuquicamata was described as a huge deposit of ore that was not amenable to the usual treatment by concentrating and smelting since much of the copper occurred as oxides. He had been working on a process for the recovery of copper from sulphide ore which involved a sulphating roast, leaching of the roasted product, and pre-



cipitation of the copper by means of limestone. Since it appeared that the ore in the Chuquicamata deposit could be beneficiated by his process, he was extremely interested and sent an engineer to Chile to investigate.

At the outset Burrage's representative was able to obtain options on the two principal properties then operating and subsequently options were acquired on numerous other claims in the district. Systematic sampling of all workings was done under his direction. Burrage first recognized the possibility of mining the entire deposit rather than just the high grade areas.

Chilex Is Formed

Since the project was too big for Burrage to finance alone, he made overtures to the Guggenheim interests, who, after a brief period of negotiation, took over Burrage's options toward the end of 1911, and on January 11, 1912 organized the Chile Exploration Co.

Exploration of the deposit by churn drilling was started by Edwin S. Berry and Walter A. Perkins in April 1912. It was soon determined that the extent and depth of the ore were much greater than anticipated.

The progress of the metallurgical engineers under the direction of E. A. Cappelen Smith, consulting metallurgist for M. Guggenheim's Sons, in the development of an economical process for extracting the copper from the oxide ore, was so successful this firm purchased the property September 1912.

Plans were made to begin large-scale development of the mine and to proceed immediately with the construction of a treatment plant. Actual copper production started in May 1915 and the operations continued under Guggenheim management until February 1923 when the controlling interest was acquired by Anaconda Copper Mining Co.

Not Standing Still

The original plant was designed to treat 10,000 tons of oxide ore daily which would yield approximately 100 million lb of copper annually. Subsequent additions to the plant, the largest of which started in 1925 and was completed in 1927, brought the capacity up to 375 million lb yearly. Additions to the plant as well as improvements in the metallurgical process, together with a higher grade of ore, brought the capacity to 490 million lb per year by 1941. In 1942, at the request of the United States Government, additions to the plant and more equipment for the mine yielded a further production of 50 million lb annually giving a total capacity of 540 million lb when operating 335 days per year. At that time it was realized that a production of this magnitude could not be maintained from oxide ore for any extended length of time, due to the decreasing grade and depletion of the oxide ore reserves. The increase, however, was justified to assist the U. S. in obtaining the copper necessary for its war effort.

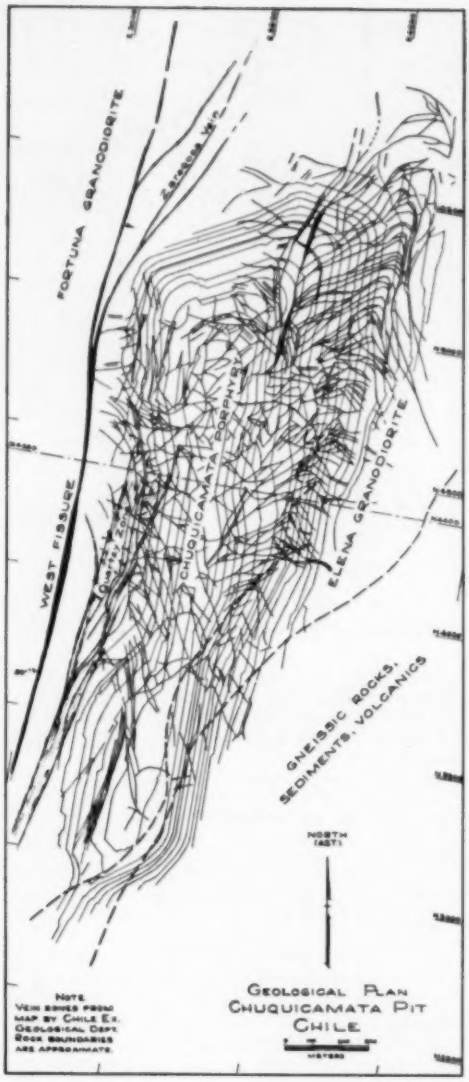
By 1948 large tonnages of sulphide ore were being exposed in the pit and were by-passed in the mining operations as far as possible. Continued removal of oxide ore exposed greater tonnages of sulphide ore which up to this time could not be treated for the production of copper as no plant was available for beneficiation of this type ore. To overcome this difficulty and still maintain a production of 540 million lb per year, the board of directors authorized the construction of a plant for the treatment of 30,000 tons of sulphide ore per day. The vast body of sulphide ore underlying the projected depth of the open pit insures a source of copper for many years to come.

Geology of the Chuquicamata Orebody

"Nature appears to have resorted to little fanfare in creating a setting for this great concentration of metallic wealth."

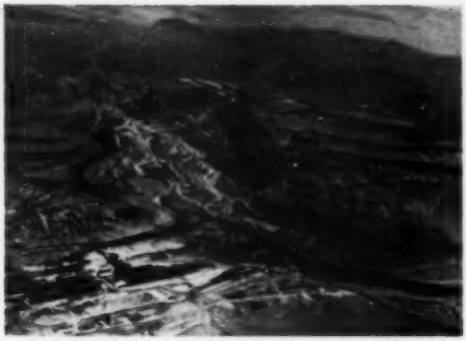
V. D. Perry,

Chief Geologist, Anaconda Copper Mining Co.

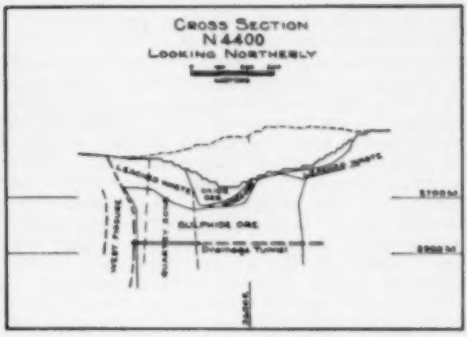


AS the traveler approaches Chuquicamata from the seaport of Antofagasta, crossing in a northeasterly direction the 150 miles of coast range and completely barren pampa with its low, rolling hills and broad depressions, the gradual rise to an elevation of 9000 ft is imperceptible, and it is hard to realize that the smoothly contoured hills east and west of the mine are from 10,000 to 11,000 ft above the level of the sea. The deceptive monotony of the scene is relieved only by the background of an unbroken line of lofty volcanic peaks, part of the Andean Cordillera, which rise, snow covered, against the eastern skyline.

The drab geographical foreground is matched by the geological monotony of unaltered, gray rock



Chuquicamata's monzonite porphyry ore zone is an elongate, dike-like body, surrounded by granodiorite and older formations. Bounded by the West Fissure on the one side, the eastern cutoff in Elena Granodiorite is gradational. Sulphide-oxide ore is irregularly distributed as unleached peaks in the oxide zone, sulphide ore is beneath.



formations that surround the great mining camp. To one familiar with the spectacular color-stained hills, countless prospect cuts and shafts, and the old mine dumps of a typical mining camp of the western United States, it is difficult to reconcile the evident lack of mineralization around the Chuquicamata pit with the extraordinary size and grade of the ore deposit. Nature appears to have resorted to little fanfare, either geographical or geological, in creating a setting for this greatest concentration of metallic wealth.

The rocks at Chuquicamata are essentially granitic and invade a complex of metamorphic, sedimentary, and volcanic formations. Localized stocks of granitic rock outcrop along a narrow belt 95 miles long in a N 15° E direction, extending from Caracoles on the south to El Abra on the north. These bodies appear to be small cupola-like expressions of an underlying batholith, formed on a vast scale along a tectonic trend of continental character and magnitude. In the locality of the Chuquicamata mine, the Fortuna granodiorite outcrops west of the pit, while a granitic rock of slightly different character, locally called Elena granodiorite, together with the Chuquicamata porphyry, are the mineralized host rocks within and immediately east of the pit.

The Chuquicamata monzonite porphyry is an elongate, dike-like body, surrounded by granodiorite and older formations. Apparent gradational boundaries between porphyry and granitic rock suggest an intimate genetic association, but late magmatic and hydrothermal alteration effectively mask the contact zones and make correct interpretations difficult and uncertain. Varieties of the porphyry may have predominant, stubby plagioclase, or may be spotted with large, well formed, pink orthoclase crystals. Quartz "eyes" are usually present and vary greatly both in size and manner of distribution; they are most prominently developed near the central core of the porphyry. Fine-grained facies often possess the appearance of aplite, studded with occasional quartz phenocrysts. Orthoclase seams and narrow, dike-like stringers of glassy quartz lace both fresh and altered porphyry and extend outward into surrounding granitic rock. Striking similarities to the typical monzonite porphyry occurrences of Ajo, Morenci, Cananea, Chino, Bingham and other camps are noteworthy and have obvious correlative significance. They tell a story of the consistent, close, genetic relation between a specific type of monzonite porphyry, with its end-stage deuteric effects, and the copper mineralization of the great porphyry copper deposits.

The Orebody

The orebody itself is confined principally to the porphyry area with a strike of N 10° E and a length of approximately 10,000 ft. In plan outline the north end is blunt, widening rather abruptly to a maximum of 3600 ft, and then tapering gradually to a relatively narrow wedge at its south extremity.

The west boundary is marked by the West fissure. It is a fault of evident magnitude, composed of numerous thick gouges and crushed zones, striking N 10° E, with steep west dip at the south end of the mine changing to a steep east dip at the north end. West of the fault the rock is fresh Fortuna granodiorite, while immediately east of the fault intense hydrothermal alteration completely masks the original rock character. Copper mineralization closely parallels the most easterly strand of the fault, but there is a significant and generally consistent band of altered rock with limited mineralization, separating the east strand and the metallized zone.

Drag sulphides within the fault gouges testify to the existence of post-mineral movement, but the general parallelism of porphyry, ore veins, and the fault strongly suggests that the West fissure was an early, pre-porphyry structure along which profound, primary crustal adjustment occurred. The evidence is strong for believing that it controlled emplacement of the porphyry, quartz and sulphides, ending its activity in final relief of regional stresses after the close of the mineralization epoch.

A 300 to 600 ft width of quartz and sericite, containing the strongest iron and copper mineralization in the orebody, parallels the West fissure and is separated from it by 50 to 200 ft of highly altered but poorly mineralized rock. Branching from it to form the eastern part of the ore zone, there is a system of strongly mineralized subsidiary fractures with northeast, east, and southeast strikes that intersect and ramify as a network of veins and veinlets, lacing altered and mineralized rock. Sales states in a private report "The Chuquicamata Orebody," March 1950; "The intensely mineralized belt constituting the orebody contains veins and innumerable criss-cross vein structures with intense mineralization and alteration of intervening porphyry. Although on a much larger scale, the structural pattern here shows a striking similarity to the Lenoard vein at Butte." The accompanying geological

plan of the pit clearly illustrates this conformity with the Butte horse tail pattern.

The oxide orebody is confined to the central and eastern part of the mineralized zone. It is composed of a thick layer of soluble copper sulphates, principally antlerite (Jarrell, O. W., "Oxidation at Chuquicamata, Chile," *Economic Geology*, 39: 251-286, 1944) with small amounts of brochantite, chalcanthite, krohnkite, atacamite, chenevixite and many other unusual copper oxide minerals. An irregular leached layer of iron oxide waste underlies the oxide orebody and separates it from enriched sulphides. The accompanying cross section shows the relation of various units of the deposit.

A reasonable and generally accepted explanation for the genesis of the oxide orebody, the waste layer and the chalcocite blanket, assumes formation of an earlier body of secondary sulphides followed by abrupt lowering of the water table due to climatic change or crustal disturbance and exposure of the early secondary sulphides to oxidation in an extremely arid climate. The secondary sulphide ore that occupied the eastern, less pyritized part of the mineralized zone, where available subordinate pyrite had been converted entirely to chalcocite, generated only weak acid, which was readily neutralized by the feldspathic gangue. Thus copper was fixed as sulphates and other salts in the oxide zone to form the present copper oxide orebody. Underlying material, originally too deep to be affected by the earlier cycle of enrichment, when exposed to oxidation above the new water table, had sufficient pyrite to supply the acid to leach primary sulphides and contribute part of the copper in the present enrichment blanket beneath the depressed water table, incidentally leaving a layer of iron oxide waste between overlying copper sulphates and underlying chalcocite.

Mixed sulphide-oxide ore is irregularly distributed as relatively tight, unleached, residual peaks in the oxide zone and as an erratic transition zone at the top of sulphides.

In the broad quartz zone forming the west panel of the mineralized belt, oxidation has removed all signs of copper, evidently because sufficient ferric sulphate was generated from high-pyritic material to thoroughly leach that metal. The host rock was principally quartz with sericite, which provided an inert gangue, incapable of neutralizing the acid water and, through its high porosity, permitted easy migration of the copper-bearing solutions to a deep zone of secondary chalcocite enrichment.

A tunnel driven approximately 1300 ft below the original surface for the purpose of draining the orebody, has furnished valuable information on the geology of the sulphide zone. Development on this tunnel level has shown that the west side of the orebody beneath the leached capping is composed of chalcocite with varying amounts of enargite, pyrite, and scattered molybdenite associated with sericite alteration. While high grade copper values are due in part to secondary chalcocite enrichment, there is evidence that the original primary mineralization contained important enargite with some chalcopyrite and that, in addition to these minerals, part of the chalcocite may have a deep-seated primary origin. Deep holes within the zone show diminishing secondary enrichment but continuity of the enargite with chalcocite at depth; they have bottomed in commercial-grade copper mineralization. At the tunnel level the sericite zone, with predominant chalcocite-enargite-pyrite, grades eastward into a zone of clay alteration, with associated chalcopyrite-covellite, that forms the central panel of the orebody. The clay zone in turn grades eastward into chloritized rock, mineralized with chalcopyrite and specularite. Bornite and small amounts of sphalerite and galena, are subordinate sulphides. Covellite has formed preferentially where chalcopyrite is the principal primary mineral and it also occurs locally in the enargite-pyrite zone close to the West fissure. While much of the covellite replacing chalcopyrite may be of secondary origin, its association with enargite appears to be primary.

Future Field and Laboratory Study

To the geologist at Chuquicamata, viewing the drab setting and lack of mineralization in the hills around the camp, the orebody itself presents a challenge and an opportunity. In it he observes the distinctive geologic features of Western porphyry copper deposits. He sees, too, the extraordinary mass of fissure veins, intersecting, branching and breaking into horsetail zones, as at Butte, Montana. Here, then, in bold contrast with a relatively unaffected rock background, exists a remarkable localization of copper ore, containing outstanding geological characteristics similar to many other great copper concentrations of the Western Hemisphere. At Chuquicamata, there is an evident focus within the porphyry mass where effects of end-stage magmatic processes, complex fracturing and hydrothermal mineralization of the highest intensity, coincide and culminate to form an extraordinary concentration of metal. Obviously, in the secret of its origin there is a master key to many problems connected with the genesis of copper ore.

The insatiable current demand for copper and the dependence of future material progress on a continuing adequate supply of the metal, call for intensified investigations in the field and laboratory on the subject of fundamental ore-forming processes. Chuquicamata contains an unparalleled opportunity for such investigation, and the geological research organization maintained by Anaconda at Butte is aiding in the Chuquicamata study as part of the general objective of developing better ways for finding new copper ore. Within and between known centers of copper production in North and South America there are critical areas partly covered, poorly prospected, and little understood where such methods, based on knowledge of the ore-forming process at Chuquicamata, may have significant future application.

Chuquicamata geology has been described by A. V. Taylor, Jr. in the 1933 Volume of *Copper Resources of the World*, International Geological Congress. Later publications by Victor Lopez, Mark Bandy, and Oscar Jarrell, together with numerous unpublished reports and maps by Waldemar Lindgren, Reno H. Sales, Walter March, Jr., Lester G. Zeihen, Glenn C. Waterman, and Charles Meyer have added a voluminous file of data and opinion. Many of these geologists, animated and directed by the original work of Mr. Sales, have devoted important parts of their professional careers to studies of Chuquicamata geology; any attempt to discuss the subject must be based in large measure on the work they have done. Mr. Waterman, chief geologist, Chile Exploration Co., is conducting a comprehensive survey and study of the Chuquicamata region including the pit and drainage tunnel; Lester Zeihen, former chief geologist, and Charles Meyer are engaged at Butte on laboratory investigation of the same problems. To all of these men, who have freely originated and generously presented many of the observations and ideas contained in the following paragraphs, grateful acknowledgment and full credit are given.

Salient Statistics, Chuquicamata Open-Pit Mine

GENERAL:*	
Continuous operation since 1915	
Total tons removed from pit	523,608,000 tons
Oxide ore	362,911,000 tons
Waste material	160,557,000 tons
Total ore remaining in pit	260,000,000 tons
Oxide ore	120,000,000 tons
Sulphide ore	140,000,000 tons
(vast sulphide reserves below pit not included)	
Total copper produced	10,212,000,000 lb
* Statistics as of Sept. 30, 1952	
MINE PRODUCTION:	
(when sulphide plant is complete)	
Total tons mined daily	105,000 tons
Sulphide ore	30,000 tons
Oxide ore	30,000 tons
Waste material	45,000 tons
BLASTING:	
Avg. tons per ft of hole (9½-in. and 12-in. diam)	48.9
Avg. tons per lb of explosive	3
LOADING:	
Avg. tons per 8-hr shovel-shift	
4 and 4½-yd shovels	3500
11-yd shovel	7000
Avg. all shovels	3900
TRAMMING:	
Avg. net tons per locomotive-shift	1325
LABOR:	
Foreign mine staff	31
Total employed at mine	1550
(incl. repair shops and 10 pct absentees)	
Total tons per man-shift	64.9
(6 mo. avg., 1952)	

Open-Pit Mining Operations

High degree of mechanization coupled with distance from source of supply necessitates extensive repair shops and large inventories.

G. S. Wyman and L. E. Fish

Mine Superintendent and Asst. Mine Superintendent, Chile Exploration Co.

CHUQUICAMATA open-pit mine is capable of producing a total of 105,000 tons daily. When the sulphide plant is operating to capacity the distribution of this quantity will be approximately 30,000 tons sulphide ore, 30,000 tons oxide ore and 45,000 tons of waste. At that time the combined capacity of the oxide and sulphide plants will be about 540 million lb of copper per year.

Twenty one benches have been opened to date and all are still active except the three top ones at the northeastern limit of the pit. These have been pushed back to final slope line. The pit is 8850 ft long, 3540 ft wide and 980 ft deep. Benches are not all developed to a standard height as the large development shovels used in cutting new benches have been of different size and range. At this time new benches 52½ ft high are being developed with an 11-yd shovel.

Ore haulage through the greater part of the pit operation has been on down grades to the crushing plant. With the opening of bench F-4 in April 1944 the first adverse-grade ore haul was encountered. Of the total tonnage now being mined, 35 pct comes from the four benches that are below the level of the crushing plant yard. Waste had been dumped on the same or nearly the same level as the bench from which it was removed up to the opening of bench F-4. Waste from this and lower levels is hauled to the level of the crusher yard where it must cross the flow of ore traffic to reach the only available dumping space to the east of the mine. The average ore haul is 3.13 miles one way shovel to crusher, and the average waste haul from that portion handled by railroad is 3.32 miles one way from shovel to waste dump. Waste that is being removed by truck haulage from the upper part of the mine is hauled an average of ½ mile. Ore loaded by trucks is hauled ¾ mile to a car loading ramp and then by rail to the crushing plant.

Drilling and Blasting

There are two types of development for blasting being used: One for opening new benches and the other for bank shooting. The former consists of 9½ in. diam churn-drill holes drilled 11.8 ft center to center in rows 10.2 ft apart with the holes staggered in the rows thus giving an equilateral-triangle pattern of 11.8 ft on a side. After the first block, 492 ft long by 71.5 ft wide, is shot in the pattern described above, the cut is advanced by smaller shots consisting of four rows of holes spaced 23 ft center to center with the rows 23 ft apart. These shots are fired in conjunction with side shots in which the holes are drilled 33 ft center to center and 29.5 ft back from the toe, thus squaring up the bank for subsequent shooting.

An average bank shot consists of about 60 holes drilled in two rows parallel to the face, 30 holes to

New York-born G. S. Wyman, mine superintendent, is proud of mine's extensive repair shops.

L. E. Fish, assistant superintendent, says old-hand Chileans make good gang bosses.

the row. The front row is located 21 ft back from the toe with 32 ft between holes. The rows are spaced 19.5 ft apart. The holes are loaded with ammonium nitrate, free running dynamite in a single charge in the bottom. Spacing and hole size are varied according to the hardness of the ground in various sections of the mine, so that enough column load is obtained to eliminate the need for deck charges.

Primacord detonating fuse is used to fire each hole and holes are connected to trunk lines of primacord so that alternate holes of the front row are timed in relation to one another and shot before the timed holes in the second row. An electric blasting timer gives 0.018 sec delay between circuits.

About 3 tons of rock are broken per pound of explosive. This reflects the hardness of the ground since it is considerably less than the tons broken per pound of explosive in many other open-pit copper mines .

The spacing used with 12 in. diam holes in the areas of easier breaking yields more ground per ft of hole than does the closer spacing used with 9½ in. diam holes in the harder areas. The average obtained from both sizes of holes is 48.9 tons per ft.

There are 20 heavy-duty churn drills in service which supply the required footage by operating only one shift per day. These machines use a 3500-lb string of tools when drilling 9½ in. diam holes and a 5000-lb string for 12 in. diam holes. Considering all holes, the churn drills average 80.9 ft of hole per 8-hr shift.

Churn-drill bits are picked up on the benches by trucks and returned to the drills in the same manner after sharpening. Sharpening and reconditioning, under the supervision of the mechanical foreman, is done in a shop especially designed for this purpose. Heating both for forging and tempering is done in a salt bath furnace while the bucking-up, shaping and gauging is accomplished with an electrically driven bit dresser.

Loading

The mine is equipped with 14 shovels, all full-revolving, with Ward Leonard control, operating from a 5000-v ac power supply. Four of these are equipped with 4-cu yd dippers, 9 with 4½-cu yd dippers and 1 with an 11-cu yd dipper. The loading rate of the 4 and 4½-cu yd machines averages 3500 tons per 8-hr shift. The 11-cu yd machine, which is used exclusively for the development of new benches, weighs 880 tons but is extremely mobile due to being mounted on automatic leveling tractors. It loads into cars located on a track 52.5 ft above the floor on which it is traveling, thus developing the full height of a new bench in one cut. The loading rate for this machine averages about 7000 tons per shift. The average obtained from all shovels is 3900 tons per shift.

In addition to the above there is one 1½-cu yd shovel which is used for casting and utility work.

Haulage

Originally the mine was equipped with steam locomotives. These were gradually replaced with electric machines until today the transportation of all ore and waste is accomplished by 27 electric locomotives and 9 motor trucks. The last five locomotive units, which went into operation during the first half of 1950, operate from third rail or trolley and in addition have Diesel auxiliaries to permit operating when away from the source of power. Six of the original electric locomotives are now being equipped with Diesel auxiliary power. The average net tonnage hauled per locomotive shift is 1325.

Five of the trucks used for the removal of waste in the upper portion of the mine have 30-ton capac-

Bank shot of 60 holes averages 3 tons broken per lb of powder, reflecting rock hardness.

ity and are powered by 300-hp supercharged Diesel engines driving through torque converters to double rear axles. The other four trucks have 20-ton capacity and are powered by 275-hp Diesel engines driving through conventional transmission to single rear axles. All trucks are giving satisfactory performance in spite of operating at 9500-ft altitude.

The ore is hauled in 575 gondola-type, standard-gage railroad cars having 70-ton capacity. Waste is handled in 30-cu yd air-dump cars having down-drop, side doors.

Communications

Locomotives and cranes are equipped with a duplex radio communication system so that the chief dispatcher can keep in contact with all units. Traffic control on the mine railroad system is accomplished through 6 intermediate stations which are connected by telephone to the chief dispatcher's office. Each local control station has a dispatcher who takes his orders from the chief in regulating train movements and who is also responsible for the manipulation of electric or electro-pneumatic switches in his area. The radio communication system, installed in 1950, has given excellent results, especially in speeding up the train movements and has proved effective in obtaining the necessary assistance in time of trouble.

Trackage

The railroad at Chuquicamata is U. S. standard gage, using 100-lb rail on main line tracks and 80-lb rail on bench tracks and dumps. The mine system has 83 miles of track of which 53 miles are electrified. The controlling grade is 3 pct and is compensated on curves. Adverse grades required for hauling from the benches below F-4 are held to 1.5 pct. When practical, a second track is laid to all benches below F-4 on a 3 pct grade for entrance of empty trains and utility equipment. In the mining area electrified lines are powered by third rail at 650-v dc. Cross ties cut from *Roble Pellin* (a native Chilean oak which is a heavy, short-grained, and holds moisture almost indefinitely) can be used without preservative treatment even in the extremely dry climate of Chuquicamata. Main-line and dump tracks are laid in conventional manner with staggered rail joints. Turnouts are held to a standard No. 7 frog wherever practical. Bench tracks which are constantly moving are laid in square end panel sections, made with 80-lb rail, canted tie plates and screw spikes. All track sections are built in the mine salvage yard under supervision of experienced men. They are loaded on flat cars and delivered to various points in the mine as needed. Sections that become damaged in the field are brought back to the salvage yard for revision. Bench tracks are generally laid, thrown, or taken up by electric locomotive cranes. Special slings are available so that track may be moved by the shovels and while this is a satisfactory operation, only a small portion of the total is moved in this manner.

Auxiliary Equipment

The auxiliary equipment used at the mine consists of electric locomotive cranes, steam cranes, bulldozers, road patrol, track shifters, truck-mounted air compressors, air-operated spreader plow, and a fleet of motor trucks. Pick-up trucks, equipped with radio, are used by the mine staff.

Three electric cranes of 45/50 ton capacity, operating either from third rail or on cable reel carrying 3000 ft of trailing cable, were built to specifications of the Chile Exploration Co. These units are equipped with regular electric locomotive control and air brakes and have the same mobility as a locomotive. All motions on the crane proper are powered by independent motors thus eliminating frictions, bevel drives, etc. Low operating costs and availability have more than compensated for the high first cost of these units. The cranes are used in the pit for track work, shovel repair, churn drill moving, and all general service crane work.

Two steam cranes are in service: A 120-ton capacity wrecker is used in emergency or for extremely heavy lifts, and a 15-ton crane is used in shop yard.

There are 9 bulldozers which are used in preparing track grades, cleanup for churn drilling, cleanup at the shovels that load motor trucks, dump maintenance, road building, and such general service as required in any pit operation.

The diesel-powered road patrol is used on haul roads to the truck-serviced shovels as well as roads that are maintained on all benches for supervision and for truck delivery of supplies and men.

Gasoline-powered track shifters in conjunction with a spreader plow are used for throwing tracks on waste dumps. This equipment makes it possible to maintain dumps with little hand labor.

Motor trucks are used for delivery of supplies such as bits, casing, and spare parts to churn drills;

Torque-converter-equipped trucks remove waste from upper benches of pit, loaded by 4½-cu yd dipper.

Most rock is moved by rail. Several electric engines have Diesel auxiliaries for operation away from sources of power.

Used exclusively for development of new benches, this 11-cu yd shovel averages 7000 tons per shift loading rate. Weighing 880 tons, the machine is highly mobile because of automatic leveling tractors. Shovel loads into cars on bench above, developing full 52.5 ft height bench in one cut.

grease truck for shovels; fuel oil truck for Diesel locomotive and tractor service; sprinkler truck for roads and trucks for general service.

Portable air compressors mounted on trucks are taken to any part of the mine where drilling for secondary blasting is required, thus eliminating the general use of air lines on the benches. These portable compressors are equipped with a manifold for attaching air hoses and jackhammers which form part of the unit. Provision is also made for carrying the necessary drill rods and detachable bits.

Repair Shops

The mechanical shops at the mine are extensive due primarily to the long distance from the source of supply for spare parts and materials. The shops do not manufacture spare parts for shovels, locomotives, cranes, bulldozers, or other equipment on a competitive basis, but because of long deliveries, many parts are repaired which under ordinary conditions would be discarded. For this same reason considerable welding is done to reclaim worn parts, which could be replaced rather than repaired.

The electrical shops are responsible for the maintenance and repair of all electrical equipment in the mine, as well as the installation and maintenance of all bench feeder cables, sectionalizing switch houses, and third rail.

Since all shop labor, both mechanical and electrical, is charged to mine operations, the large shop forces reflect in the tons per man-shift and should be considered when comparing labor efficiency figures with other operations.

The foreign staff at the mine consists of 31 men. The average number of men employed, including the shops, is 1550, of which 280 are *empleados* and 1270 are workmen. *Empleados* are designated as white-collar workers, Chilean supervisory personnel, instrument men, draftsmen, locomotive engineers, and chauffers. The total of 1550 men covers about 10 pct that are absent for illness, vacations and layoffs. The tons loaded per man-shift average 64.9 for the year to date.

Higher standards of safety are ever the goal in all operations at Chuquicamata and excellent progress has been made by all departments in averting lost-time accidents. The mine dept. won The C. F. Kelley Award for Safety for the years 1950 and 1951. This award is presented annually to that mine in the Anaconda group which makes the best record in reducing lost-time accident percentage.

General view shows new sulphide plant, concentrator in foreground, smelter in distance at left.

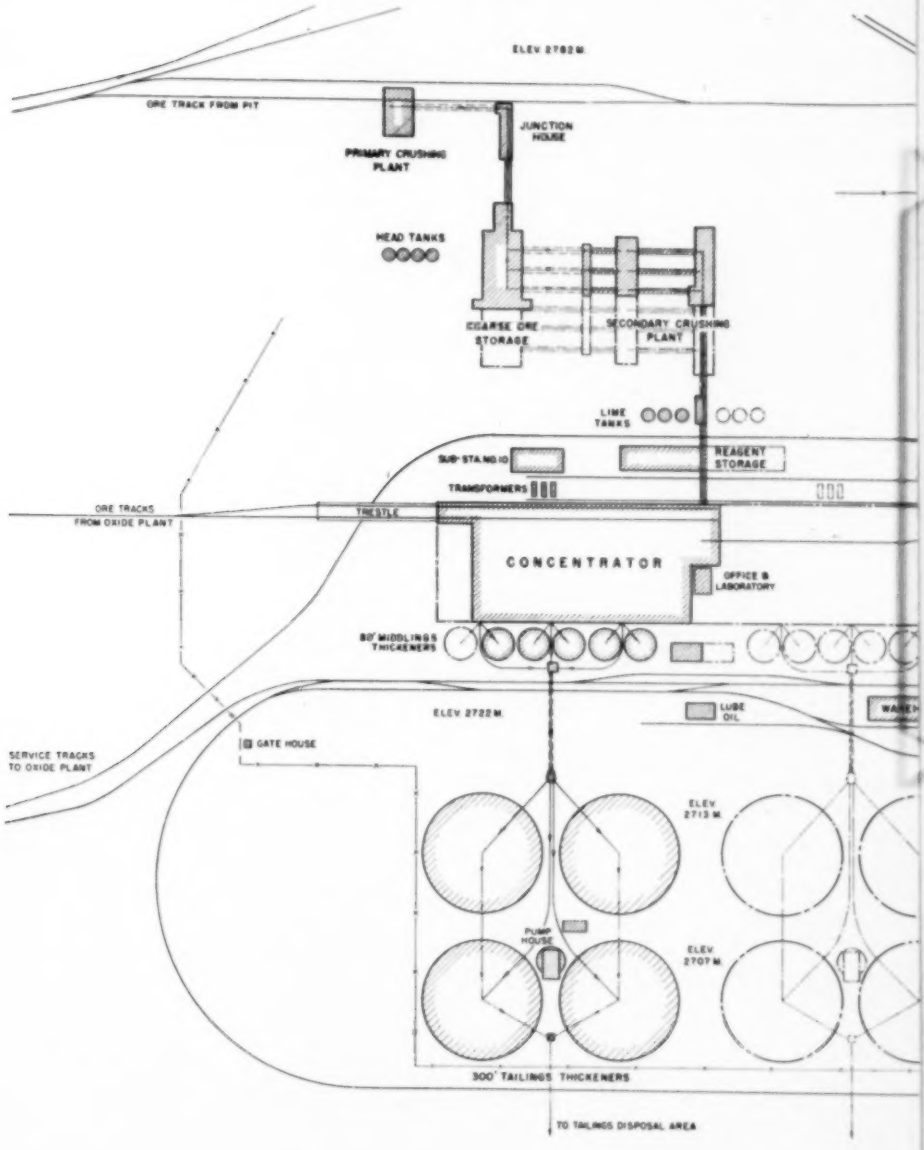
Chuquicamata

Open-pit mine

Sulphide Plant

BEFORE the new sulphide plant went into operation late this year, only the oxide copper ore from the Chuquicamata orebody was processed into copper. This has been mined since 1915, but as mining progressed in depth, mixed oxide-sulphide ore was disclosed as well as the mother lode—a vast sulphide copper orebody of undetermined depth. Sulphide ore was by-passed or stockpiled for the day when a treatment plant would be built. The mixed ores were leached of oxide copper and the residue, containing sulphide minerals was stockpiled.

Today, the giant new sulphide plant is in operation and is expected to treat up to 30,000 tons per day of sulphide ore. The oxide plant will continue to treat oxide ores for many years to come but at reduced rate.

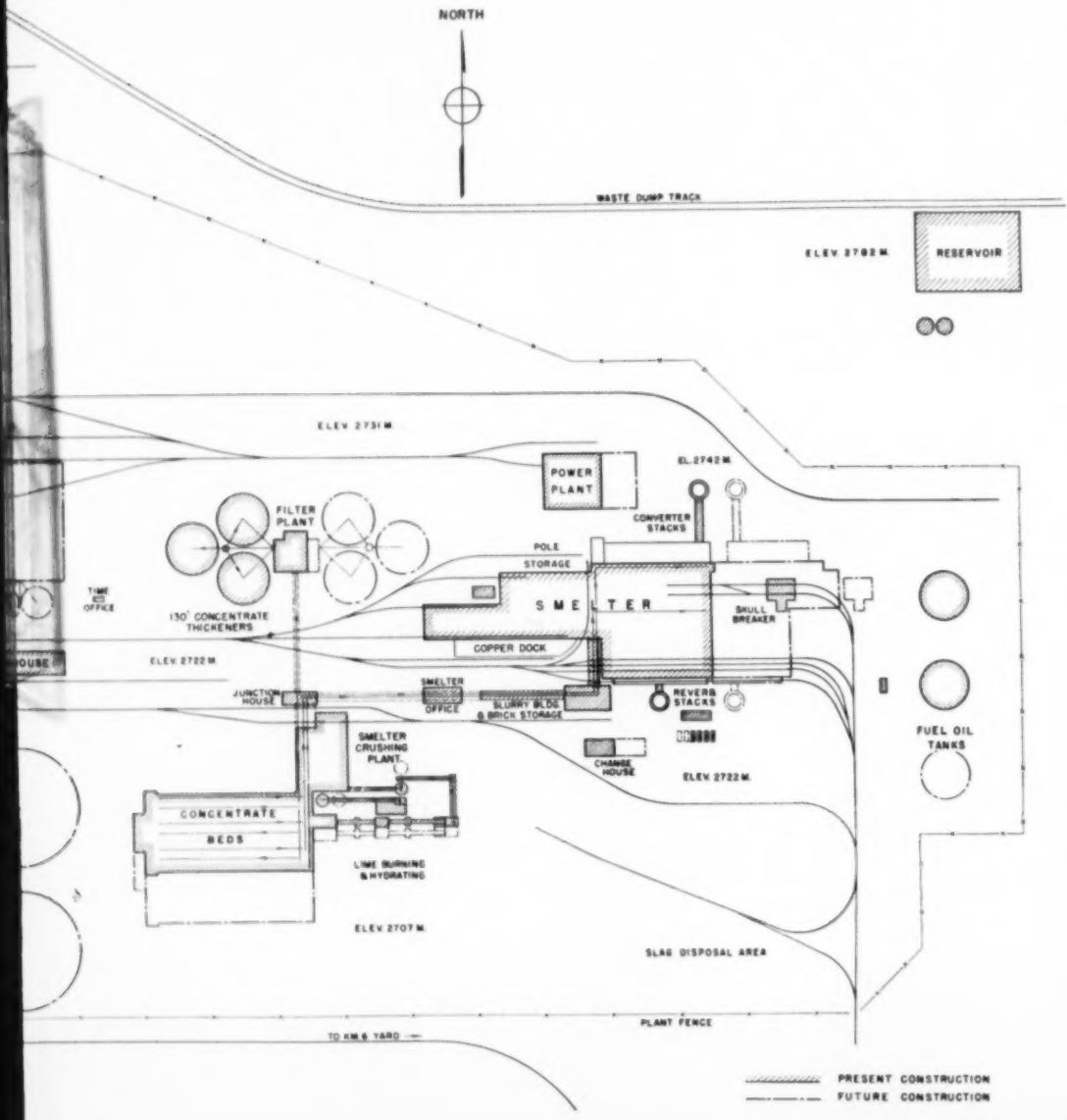


Partial view of Chile Exploration Company's open-pit copper mine at Chuquicamata. Over 500 million tons of oxide ore and waste have been removed since 1915. The new sulphide plant is now beginning to process the vast reserves of sulphide ore, and, when operating fully, will treat 30,000 tons of ore per day.

Sulphide Plant Flowsheet

Sulphide ore from the open-pit mine averaging 1.75 pct Cu arrives by rail at the crushing plant where it is reduced to ¾-in. size and transported to concentrator storage bins. The concentrator consists of 10 sections. In each section ore is reduced to 10 mesh in one 10x14-ft rod mill. The product goes to a 10x12-ft ball mill in closed circuit with two 54-in. spiral classifiers between which the ball mill product is split. Classifier overflow, 14 pct on 65 mesh goes to flotation. For regrinding, the rougher concentrate is thickened and pumped to a 6x12-ft ball mill in closed circuit with a rake classifier. Flotation tailing is thickened to 45 pct solids and flows by gravity to tailing pond. Overflow of the regrind-circuit classifier, 90 pct —200 mesh, goes to flotation. Concentrate is thickened and filtered. Cleaner flotation underflow returns to the rougher circuit. Filtered concentrate, averaging 40 pct Cu, and representing a 91 pct recovery of the sulphide copper is conveyed to the bedding plant. The concentration ratio is approximately 20 to 1.

Silica, lime rock, dust, and reverts are mixed at the bedding plant with concentrate. This mixture, averaging 37 pct Cu is charged to a 30x125 reverberatory furnace at the rate of 600 tons per day. Matte averages 45 pct Cu, 25 pct Fe, and 25 pct S and is tapped into 325-cu ft (50 tons matte) ladles from which it is poured into the 13x30-ft converters. Blister copper, 99.5 pct Cu, is transported in same ladles to holding furnaces for casting. The molten converter slag is poured into the reverberatories. Net result: 450 tons copper per day from sulphide ore.



General Design Sulphide Ore Plant

Plans achieved three key aims — simplicity — flexibility — expandability — and were executed on terrain carved to fit the job.

Wilbur Jurden

Chief Engineer, Anaconda Copper Mining Co.

THE writer's first experience with a nonferrous reduction plant of great magnitude was at the Washoe reduction works of Anaconda some 35 years ago. Here was a plant which had been planned with remarkable skill and foresight considering the time and the state of development of copper-plant practice in the year 1902. The designer utilized topography to fullest extent to provide proper sequence of operations and, what is most remarkable, to leave adequate space for future developments, most of which at that time were unknown. However, the practice then was to locate the various units of the reduction works at the most advantageous points of the existing terrain with little regard for tramming or other auxiliaries and then connect these various units by the essential trackage, conveyor systems, piping, etc., as the need developed. This occasionally led to undesirable track arrangements, sharp curves, and steep grades, especially when it became necessary to extend various portions of the plant. Conveyor systems also became rather complicated, running as they did at various angles, and such items as piping and electrical distribution were often found to be in the wrong place, entirely inadequate in size, or awkwardly arranged for any kind of extension.

This condition was not peculiar to Anaconda, for all copper plants at that time were built in the same manner and it was the constant association with these difficulties which, in the year 1925, influenced the layout of the Andes Copper Mining Co. plant. In that plant all trackage was laid out straight and level, all conveyors at right angles to each other with minimum length and number of transfers. All buildings were placed parallel and the main structures were complete for all purposes so that auxiliary buildings and *dog houses* would not be added later. Piping and electrical work was provided for in the original layout and carefully designed to avoid additions and alterations, and careful study given to every movement of material throughout the entire plant so that it would be accomplished with the least possible effort.

Naturally it was hardly expected to attain all these objectives perfectly but our efforts did succeed in creating a plant which was unique and outstanding for its time—1927. It was also most gratifying to find that these design principles contributed to considerable savings starting right in the drafting room, carrying through the construction and ultimately yielding savings in operations and manpower. Not only that, but such a plant gives the observer an impression of symmetry and order, is more attractive to the workmen, and unquestionably eliminates many accident hazards. However, the Andes plant buildings were fitted to the existing terrain instead of having terrain created to fit the buildings—an item which we found advantageous to correct on the next large plant.

At Morenci in 1939, all of the desirable features of the Andes plant such as parallel buildings, etc., were incorporated; but we went one step further—power shovels were brought in to make the terrain fit the reduction works. The result at Morenci is well-known and needs no elaboration here, but the success achieved by the design methods used for this and previous plants naturally influenced and guided the layout of the Chuquicamata sulphide plant which is the largest yet conceived.

Chuquicamata Plant Design

At Chuquicamata several factors not encountered previously complicated the problem to a great extent. The most desirable location for the smelter would allow smelter gases to blow directly into the open-pit mine already producing 60,000 tons of oxide ore per day and employing 1550 men. This, of course, would be a serious condition and, therefore, we were forced to move the smelter to a less desirable location but followed our previous experience at Morenci and made the terrain fit the job.

The most difficult problem, however, was the provision for receiving various types of ore both by rail and conveyor. These consisted of: 1—Sulphide-bearing residue from the stockpile from which oxide copper had previously been leached. 2—Sulphide-bearing residue coming direct from the leaching vats. 3—Sulphide ore crushed at the existing crushing plants and hauled to the concentrator in cars. 4—Sulphide ore from the new crushing plant adjacent to the concentrator. 5—Sulphide ore obtained

LEFT—From the air, the Chuquicamata plant appears to sprawl, but new sulphide plant has been dovetailed to receive ore from oxide-plant crushers, stockpile of leached ore and direct from leaching vats, from new sulphide-plant crusher and from future underground mining.

RIGHT—Situated to keep smelter smoke from pit, new plant is on site prepared for economic handling of all materials.

from underground mining (future) and crushed in the new crushing plant.

To summarize the foregoing, the problem consisted of locating the concentrator so that it could be supplied efficiently with ore from the existing crushing plants and ore from the new crushing plant being constructed adjacent to the concentrator.

Referring to the layout sketch (see insert and figure above) it will be noted that the principles of straight trackage, parallel buildings, etc., have been applied to the Chuquicamata sulphide plant. The layout also shows the ore track from the existing plant which travels to and over the concentrator ore bins. It was this track which established the elevation of the top of the bins, and formed the starting point from which the entire layout had to be developed. Unfortunately, the slope of the terrain was not sufficient for the fall required by the concentrator and considerable excavation was necessary. It is also noted that the tailing dewatering tanks had to be placed at some distance from the concentrator where the ground would be low enough for gravity flow.

On the smelter side, the location of which was dictated by the travel of the smelter gases, the original terrain was rough and considerable heavy excavation was necessary to establish the proper elevations for the various units.

Designed for Continuous Operation

Aside from these two features we were able to complete the layout without any extraordinary expense and have incorporated all of the features which have been proven economical and desirable in previous plants together with numerous improvements. Ample space has been provided to allow this plant to expand for many years and the layout provides for all expansion to take place without any interference to existing operations at any time. This is an important feature and one which can only be provided for by proper layout and careful attention to design in all departments.

For the initial operations, all ore will be hauled into the plant by rail. Later a conveyor system from the crushing plant will be installed permitting ore delivery by conveyor and by rail simultaneously. The crushing plant itself will originally receive ore by rail and later will receive ore both by rail and from underground. These installations will be made without shutdown. This holds true throughout the entire plant, the layout making it possible to install one or more grinding units, one or more reverberatory furnaces, one or more converters or any other equipment either in the concentrator, the smelter, or the power plant without any interference to operations being carried on at that time.

The entire layout in its present form gives one the impression of simplicity and logical arrangement of the various units. The solution of such a problem and the proper arrangement of the various units are far from being simple matters. The development of a layout, which in its final form appears to be simplicity itself, is only completed after deliberate and diligent effort has been expended and one layout after another discarded until every objectionable feature has been eliminated if it is possible to do so. In order to develop the present layout which appears to be the natural one for the job, at least 50 plans were made and discarded; most for major faults and some for minor ones. These layouts also are under the constant supervision of the various department heads as they are being prepared, and every possible objection is brought out and discussed during their preparation.

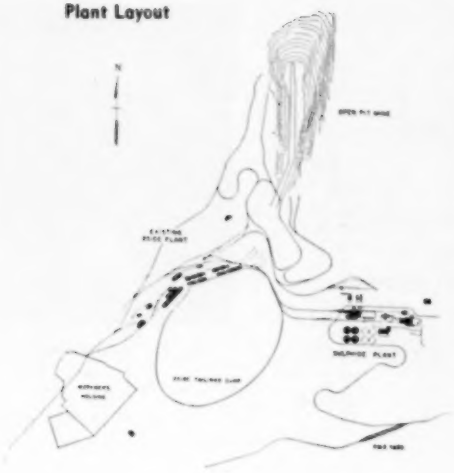
Proper design of a plant of this magnitude is so extremely important, not only in saving design and construction costs but in reducing labor and subsequent costs of operation, that no effort should be spared to make it as nearly perfect as possible.

Procurement and Control

The effort required to initiate and complete such a project as the Chuquicamata sulphide plant is difficult to visualize unless one is familiar with the various steps that have to be taken and has some idea of the essentials pertinent to a plant of this nature.

The plant is designed for and, if conditions warrant, can be expanded to consist ultimately of 24 grinding units having a nominal capacity of 60,000 tons per day, 8 reverberatory furnaces and 9 converters, of which 10 grinding units, 4 reverberatories and 4 converters are being installed under the present program. This is about 40 pct of the plant which could be constructed on this site. The quantities for this 40 pct presented a major problem of procurement, transportation by rail and water, passing through customs and final erection in a desert country at high altitude more than 5000 miles away from

Plant Layout



the source of manufacture. The tonnages shown in the following are for the present plant and will give some idea of the magnitude of the project:

Excavation	2,500,000 cu yd
Fill	2,500,000 cu yd
Reinforced concrete	152,000 cu yd
Structural steel & bins	35,000 tons
Reinforcing steel & mesh	7,500 tons
Special brick & mortar	6,350 tons
Railroad track & fittings —17 Miles	3,350 tons
Railroad cars & locomotives	6,700 tons
Piping air, oil, water, reagent, concentrate and tailing	20,000 tons
Electrical equipment, control & cable	12,150 tons
Mechanical equipment	40,450 tons
Construction equipment, tools, etc.	2,700 tons
TOTAL	134,200 short tons

To transport this material from New York to Chilean seaport a total of 408 shipments by ocean freight were made, and the transportation from seaport to plant site, a distance of 163 miles, required the hauling of nearly 8000 loaded cars on the metergage railroad. In addition to this, all cement (225,000 bbl), lumber and other items produced in Chile were purchased there, transported by boat and hauled to the plant over the same railroad.

Drawings and/or requisitions for every item included in these tonnages had to be completed in the New York engineering office, with a total of over 3000 drawings and 3430 requisitions, each requisition containing from several to as high as 600 items. In addition to this, there were approximately 15,000 manufacturers' drawings of structural and mechanical details. When one stops to consider that each and every item entering into this huge project must not only be designed and specified but, after purchase, must be meticulously expedited by personal contact until the expediter sees the equipment loaded on the boat for South America, one can begin to grasp the magnitude of such a project and the absolute necessity of system and efficiency throughout every division of the engineering department.

In addition to this, however, it is essential that the construction staff in Chile be kept fully informed of the status of every item and when these items arrive they must be expedited through Customs, transported to the plant site, properly warehoused and stored and finally installed in their proper position. All of the data required by the construction staff must emanate from the New York engineering office and without systematic handling of all orders, requisitions, drawings, etc., the inevitable result would be utter confusion causing serious and costly delay. However, it is unfortunate that even the most systematic regulations are of no avail when influenced by such items as the difficulty of procuring material due to defense projects, strikes, either general or in isolated plants (the latter being particularly troublesome), the loss of equipment in transit, thievery and malicious damage. All of these had to be contended with to an exceptional degree during the life of this project and each has had its adverse effect on orderly procedure, uninterrupted progress, and efficient construction. In spite of all these obstacles, however, the first units of the plant have been successfully put into operation and the remainder of the construction is rapidly nearing completion.

Superintending Chilex's sulphide plant is E. D. Tierney, big but fast moving and always available for consultation.

L. R. MacLeod, mechanical engineer, thoughtful, quietly humorous in spite of job ironing out wrinkles in new plant.

Chuquicamata Sulphide Plant:

Crushing Section

Stepped-up sulphide plant tonnage and complex classification problem with three kinds of open-pit ore — sulphide, oxide, and mixed — necessitated new crushing plant.

A. P. Svenningsen

Chief Design Engineer,
Anaconda Copper Mining Co.

IN the early stages of design it was not considered necessary that separate crushing plants be built for the new sulphide concentrator and smelter until sometime in the future. The plan was to use the existing crushing facilities for both oxide and sulphide ore. A few additions were contemplated for the existing plants, such as increased bin capacity, and possibly two new secondary crushing units.

The more the problem was studied and discussed with the plant operators, the more it became evident that it was complex. It involved the classification of different kinds of ore from the open pit mine —sulphide, oxide and mixed—and how best to separate them so that each kind of ore was given the proper processing and treatment. It also involved the problem of keeping the different ores from being contaminated in bins, hoppers and chutes. Added to these, transportation became complicated and would involve additional handling and loading of ore from crushing plants to conveyors, to bins, and finally to railroad cars which were to be hauled to the concentrator and dumped into the fine ore bin.

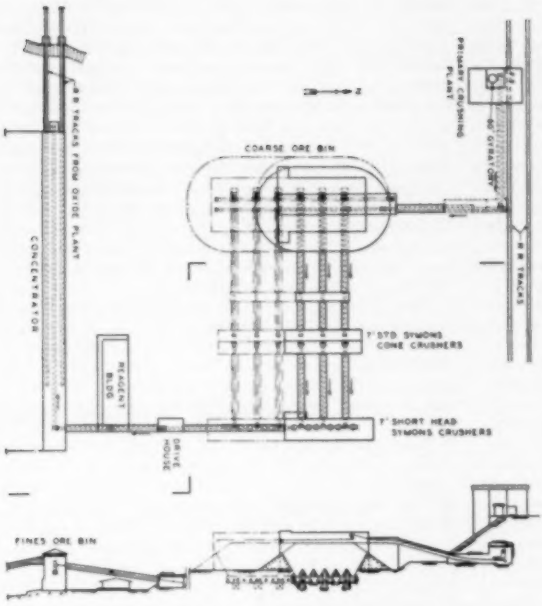
General

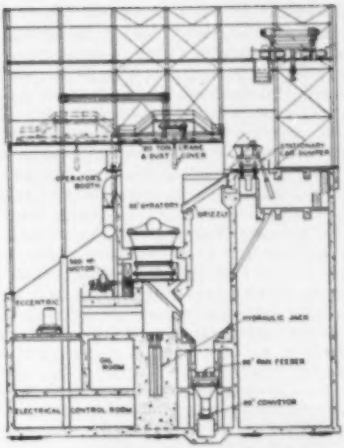
In the early part of 1951 it was decided that the concentrator be constructed with ten grinding units instead of seven as originally authorized. The smelter was to be increased proportionally and naturally also the overall tonnages of ore to be handled by the new sulphide plant. Due to this increase in plant capacity and the larger tonnages involved, the difficulties which would arise by using the existing crushing plants were increased to a point where it became evident that the building of new crushing plants for sulphide ore exclusively was technically, as well as economically, advantageous. Authorization was, therefore, given by the company to construct new crushing plants to handle 30,000 tons of ore per day, and capable of reducing the run-of-open-pit ore to the proper size feed for the 10x14-ft rod mills in the concentrator. The ore, mined in the open pit, sometimes comes in pieces as large as 6 to 7 ft diam. The rod mills may call for ore crushed to $\frac{3}{4}$ in.

The large size of ore delivered from the open pit determined that a 60-in. gyratory crusher be used as primary breaker. Such a crusher will have a capacity considerably in excess of 30,000 tons per day. The crusher will be a single discharge unit driven by a 500-hp electric motor through a tear coupling and a floating shaft. This type of drive has proven successful at a number of other crusher installations which our company has operating in the United States, Mexico and South America. The tear coupling will protect both the crusher and motor against damage in case of overload. No new features are incorporated in the design of the crusher itself, except that the discharge chute is made the full width of the crusher with parallel sides instead of the usual converging sides. This change in detail should eliminate a feature which has been a bottleneck in some of the operating plants and has caused loss of production due to ore hanging up and blocking the chute.

The secondary crushing plants will have three 7-ft standard Symons cone crushers and six 7-ft short head Symons crushers. Between the primary and secondary crushing plants a coarse ore bin will be constructed with a nominal draw-off capacity of 30,000 tons of ore. The standard Symons and the short head Symons will be in separate buildings.

All the crushing plants and the coarse ore bin are interconnected with conveyor belts for transporting the ore to the crushers at the tonnage rate desired. The final product of the new crushing plants is produced by the short head crushers. It will be delivered onto a conveyor belt leading to the top of the fines ore bin in the concentrator. A separate conveyor belt running the full length of the fines ore bin and provided with a movable tripper of rugged design will discharge the sulphide ore into the bin. The concentrator bin is planned and designed so that the installation of this additional conveyor will not interfere with the operation of the two railroad tracks on which crushed ore is brought from the existing oxide plant. Thus when completed the bin can be filled simultaneously by ore from the new crushing plant and by ore from the existing leaching plant.





LEFT — Plant layout is unusual in that main floor of crushing plant is approximately level with natural ground, little excavation being required. A 100-ft fill on three sides carries the ore track connection with the open pit. ABOVE — Novel feature of gyratory unit is combination 20-ton crane and dust cover.

A cross-section showing the principal features of the primary crushing plant is shown on the sketch above.

Car Dumping

Sulphide ore will be brought to the primary crushing plant in trains made up of eight or more side dump railroad cars. Each car will have a nominal capacity of 40 cu yd or approximately 90 tons. Inasmuch as it is the intention to use these cars for transportation of ore only, it is not necessary that the cars be completely self-contained with air cylinders for dumping. This latter feature is employed in mines where the cars are used alternately for ore and waste. In our case the cars designed entirely for ore transportation will be dumped by a stationary dumper built into the primary crushing plant. This dumping device is hydraulically operated by a system using a maximum pressure of 1000 psi. The speed of dumping can be regulated from the operator's booth and the dumping can be interrupted at any position desired, thus giving the operator complete control to pour ore into the crushing plant at the rate which he considers best. He can also stop feeding ore entirely in case there is any indication that a large piece of ore threatens to choke the crusher.

The head of the dumping device which engages the cars is so designed that it will grip the cars and lock itself to the car after it has lifted it a slight angular distance. This device, which is simple and has no moving parts, will make it impossible for the car to get out of control and tip over and fall into the crusher.

The ore which is discharged by the dump cars passes directly over a grizzly. This feature is novel. All existing crushing plants handling ore as big as the Chuquicamata open-pit ore, are designed to dump into an ore pocket from which the ore slides onto the grizzly. We do not consider an ore pocket necessary in a plant where the discharge from the cars is so completely under the control of the operator as it will be in the new crushing plant. However, the grizzly is of extra heavy design and a manganese steel apron is placed over the upper part of the grizzly to protect it against impact of large pieces of ore. A Ross-type chain feeder can be installed above the grizzly in case the actual operation of the plant indicates that this would be desirable for retarding the ore sliding over the grizzly into the gyratory.

An improvement in design of the installation of the crusher proper is that the floor which surrounds it at the top, and which supports a pocket of ore, has been lowered approximately 6 ft. It was found in operation of some of the company's existing crushing plants that this floor was a source of constant trouble, requiring reinforcing and rebuilding every so often, when it was damaged by the pounding of the dropping ore. This has been costly and time-consuming. In the new design the floor was, therefore, lowered and made a very thick concrete slab, heavily reinforced and supported by walls resting on the crusher foundation. A horizontal seal prevents any possible leakage of dust into the room surrounding the lower part of the crusher. Access to the crusher discharge chute has also been provided to facilitate chute inspection and relining.

Operation and Dumping

A 20-ton crane is provided for clearing jams in the primary crusher. It will serve a dual purpose as it has been transformed into a complete dust cover for the top of the crusher enclosure. The walls on three sides are carried up to the level of the supporting rails. In the above illustration, the crane is shown in the closed position and the open position is

dotted in. All the crane controls are located in the operator's house. When it becomes necessary to reach the crusher with the 100-ton hook of the general purpose crane, the 20-ton crane and dust cover is moved out of the way so as to leave the entire crusher enclosure open at the top. This operation will be needed for general maintenance purposes or when a major repair is required.

The operator's booth is located near the top and on the side of the crusher enclosure opposite to and facing the car dumper as shown on the drawing. The operator will have a clear view of the side dump car and of the ore as it passes from car to grizzly and then into the crusher bowl. Very strong lights placed near the top of the enclosure will provide illumination and particularly spotlight the top part of the crusher so that the movement of the ore will be visible at all times. From his station the operator will be in full control of the dumping of the ore cars—the travels of the bridge and trolley of the 20-ton dust cover crane—the lowering and raising of the special hook for breaking jams—the starting and stopping of the 60-in. gyratory and the starting and stopping of the dust control system. Indicating lights registering the performance of all the mechanical equipment in the crushing plant will also be located in the operator's booth. The front end of the booth will have a large glass window to give the operator an unimpeded view of the crusher, grizzly and dump car. This window is designed so that it is easily cleaned on the crusher side when necessary. The booth will be air-conditioned.

Our crusher operators have found that frequent inspection of the eccentric will often prevent costly and time-consuming repairs. The new primary plant will, therefore, be furnished with a hydraulic jack for lowering the eccentric assembly so that it can be inspected conveniently or replaced if necessary. This work can be performed speedily in a dust free atmosphere by the arrangement developed and shown in the illustration. The simple design is possible only because the crusher is a side discharge machine. The eccentric is handled on a small truck running on rails, which brings it into a position where the main crane can lift it.

Plant Layout

The lowest level of the primary crushing plant has access openings to a road, thus permitting trucks and passenger cars to enter the plant proper at this level. This arrangement will afford convenient delivery of spare parts, supplies and equipment of all sorts. The electrical control room requiring an area of 1450 sq ft is located at this level. The transformer substation is located immediately outside the control room. This also covers an area of 1450 sq ft. The equipment in the control room will consist of all primary controls for large motors in the primary crushing plants and those needed for driving conveyors between this plant and the coarse ore bin. It will also include generation and control of direct current for variable speed drives, magnets and magnetic pulleys and overhead cranes.

In one respect the design and construction of the new primary crushing plant differs from all other gyratory plants of this size in the U. S. or South America. The natural ground is approximately level with the ground floor of the crushing plant. Very little excavation will be required but the crushing plant rests on firm and solid rock. Existing plants of this size have all been set deep in the ground requiring excavation up to 90 or 100 ft deep. In our case the crushing plant structure will be completely above the ground and a fill slightly less than 100 ft high will be placed around three sides of the structure. This fill will carry ore tracks connecting the plant with the open pit. The top of the fill corresponds in elevation with that of tracks at the open pit exit, and also with that of one of the mine waste dumps. The unusual arrangement presented the designers with a number of structural problems applying not only to the crushing plant itself, but also to the tunnels for the conveyors connecting with the coarse ore bins. The conveyor tunnels must support the weight of the 90-ft embankment and required extra heavy walls and roof.

The design of the coarse ore bin is kept as simple as possible. In the plainest terms, it is a large pile of ore having a super-structure supporting the conveyors feeding ore to the bin and a concrete substructure with twelve openings through which the ore is drawn by means of 48-in. pan feeders onto the conveyors leading to the secondary crushing plants. Each of the three 7-ft Symons cone crushers will be fed by a 48-in. belt conveyor. It is of the utmost importance that the rate of feed to these crushers be constant and continuous. The ore will, therefore, be fed from the coarse ore bin to the conveyors in a steady stream by variable speed manganese steel pan feeders. Four pan feeders are provided for each crusher to insure a continuous supply of ore to the Symons.

The foundations for the 7-ft standard, as well as for the 7-ft short heads, are placed on the same level as the yard surrounding the two crushing plants. Each plant will be provided with a 75-ton overhead traveling crane. When a crusher needs overhaul, repairs or relining, the entire machine is removed from its foundation by the crane and placed on a suitable support in the repair section. Complete spare standard and short head crushers have been purchased and are held in readiness in the repair bay to take the place of any crusher removed. This is a feature which was introduced in the Cananea Consolidated Copper Co.'s new secondary crushing plant built in 1944. The operators found this feature most desirable and time saving.

Lubrication

In the primary crushing plant a lubricating system will be provided for the 60-in. gyratory. Likewise, independent lubricating systems will be installed in each of the two Symons plants. The three systems are similar and differ principally in the quantity of lubricating oil supplied to the machines. The systems are designed for gravity feed. Head tanks are installed at suitable elevations in each crushing plant. Sump tanks are located in pump rooms below the crushers to receive the oil which drains back by gravity. From these it is pumped through coolers back to the head tanks and, when necessary, to purifiers. The water seals provided on the Symons crushers may overflow if not carefully controlled, thus contaminating their lubricating oil. For that reason, the secondary plants have been provided with a common centrifuging station where the oil is cleaned and any entrained water is removed. Oil is cleaned by a pressure filter at the primary crushing plant as the 60-in. gyratory has no water seal.

The oil piping to all crushers has temperature and flow devices to automatically protect the machines

against any lubrication system failure and to indicate such failure by alarms.

The bearing in the spider of the 60-in. crusher is lubricated from a separate system. The lubricant is a very heavy oil pumped into the bearing under high pressure. No return lines are used for this oil.

Dust Control

As a result of experience gained at the existing crushing plants regarding the value of effective dust control systems, automatic bag type dust collectors are being provided at the new primary and the two secondary plants. All units will be sized to allow ample areas of filter medium even during the shaking periods when some bags are out of service.

In the primary crushing plant the dust filters are located inside the building. The exhaust fan has a capacity of 65,000 cfm. Under normal conditions the fan will pull air through the opening above the grizzly, where the side dump cars discharge the ore, at a velocity of approximately 100 fpm. This indraft will prevent any dust, liberated or created by the dumping of the cars, from spreading into the air and becoming a nuisance. Likewise, air is drawn at the discharge end of the 96-in. pan feeder where dust otherwise would be created and spread into the crushing plant. Similar arrangements are provided for the secondary crushing plants and at all transfer points from one conveyor to another. The dust control systems for the secondary plants are located outside the buildings proper.

The crushing plants are now being constructed and it is expected that operation of these new facilities will start during the summer of 1953.

Chuquicamata Sulphide Plant:

Concentrator Design

Sectionalized design provides flexibility on site where gravity flow minimizes pumping. Flotation and grinding tests under actual operating conditions determined most of major equipment before final layout was chosen.

E. F. Roffo
Design Engineer,
Anaconda Copper Mining Co.

THE design of the Chuquicamata concentrator offered an unusual combination of problems, all of which had, in one way or another, a definite effect upon the final arrangement of all the equipment and necessary facilities.

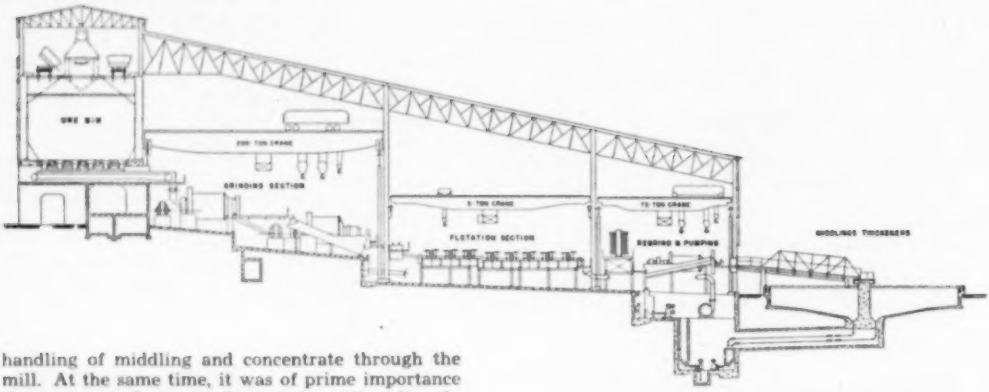
First of all, a remotely located project of this magnitude required careful selection of equipment with an adequate supply of replacement parts and in some cases necessitated stand-by spare equipment ready for immediate use. Fortunately, in the course of determining the proper flowsheet, in itself a prime factor in the concentrator design, it was possible to conduct numerous grinding and flotation tests at the Anaconda and Andes concentrators. As a result of these tests made under actual operating conditions, most of the major equipment best suited for treating the Chuquicamata ore efficiently was decided upon while preliminary design layouts were still being made.

The design has been based primarily on the treatment of sulphide ore crushed to about ¾ or 1 in. However, until the new sulphide crushing plant is completed it will be necessary to handle ore crushed to ⅜ in. in the existing oxide crushing plants and to some extent a mixed ore which has had its oxide copper content removed by leaching. This mixed ore, known as leached residue, will contain a certain amount of sulphuric acid from the leaching process which necessarily requires special protective measures.

Several factors influenced the final location of the Concentrator. The only established fact known during preliminary design layout was the elevation of the dumping floor over the fine ore bin. This was predetermined by the desired level railroad grade across part of the existing tailing and leached residue dumps from the oxide plant to the bin.

Next it was necessary that a site be chosen that would not require an excessively long haul from either the present oxide plant or the mine and in addition provide a fairly constant natural ground slope sufficiently wide to accommodate the length of the ultimate concentrator. The site as finally chosen to meet the foregoing restrictions provided a natural ground slope of 6 to 7 pct. This was excellent, especially in a region subject to earthquakes, as it assured that foundations of heavy equipment and footings for building and crane columns would rest on cut or undisturbed ground. However, this shallow slope, although having some definite advantages in the overall consideration of plant layout, resulted in the need for extensive studies in the general arrangement of equipment. Wasteful and excessive vertical drops had to be eliminated in the

Section Through Concentrator



handling of middling and concentrate through the mill. At the same time, it was of prime importance that economical gravity flow of the tailing directly to thickening tanks was assured. Of course, as in all concentrators, some pumping was necessary. Those pumps that were required handled only the smaller quantities and ultimately, as the design developed, made it possible to obtain a higher degree of flexibility. Another problem, and perhaps the most difficult, was the necessity of providing an arrangement of equipment with adequate auxiliaries, all made up of semi-independent sections, whereby it would be possible to increase the original mill capacity by definite increments without affecting existing facilities and operation. This problem was two fold. Because of the uncertain economic situation, the size of the initial mill was changed on several occasions during the design stage. In addition, a sectionalized design provided the flexibility needed for any future expansion program.

Ore Bins and Feeders

A final problem involved the methods of ore delivery to the fine ore bin. The leached residue, as well as the sulphide ore crushed at the existing oxide plant, is hauled by rail in 60-ton air-operated side dump cars over the bin. Two tracks spaced 31 ft 6 in. apart extend the full length of the bin, with each track having a third rail for electric power. Upon completion of the new sulphide crushing plants located immediately north of the concentrator, ore will also be delivered by conveyors with a traveling tripper operating along the center of the bin between the ore tracks. Two continuous dumping slots with removable covers were designed to accommodate both the tripper and ore cars dumping towards the center of the bin.

The design of the ore bin was based on an approximate drawable storage of a nominal day's supply totaling about 2500 tons of dry ore per section. A width of 49 ft was determined by the clearances needed on the dumping floor to accommodate two tracks with a conveyor between. The length was established as 45 ft per section by overall study of the equipment requirements for each grinding section. Consideration was given to various types of bins and after careful study and investigation it was decided that a rectangular bottom bin having replaceable vertical steel walls would be the most suitable and the easiest to protect from acid. Asphalt mastic lining was placed on the bottom and along the sides where it was not subject to wear by moving ore. Further precautions included sloping the bin bottom to a series of lead pipe drains to remove any possible collection of acid, and the use of acid resistant stainless steel to line the feeder openings in the bottom of the bin.

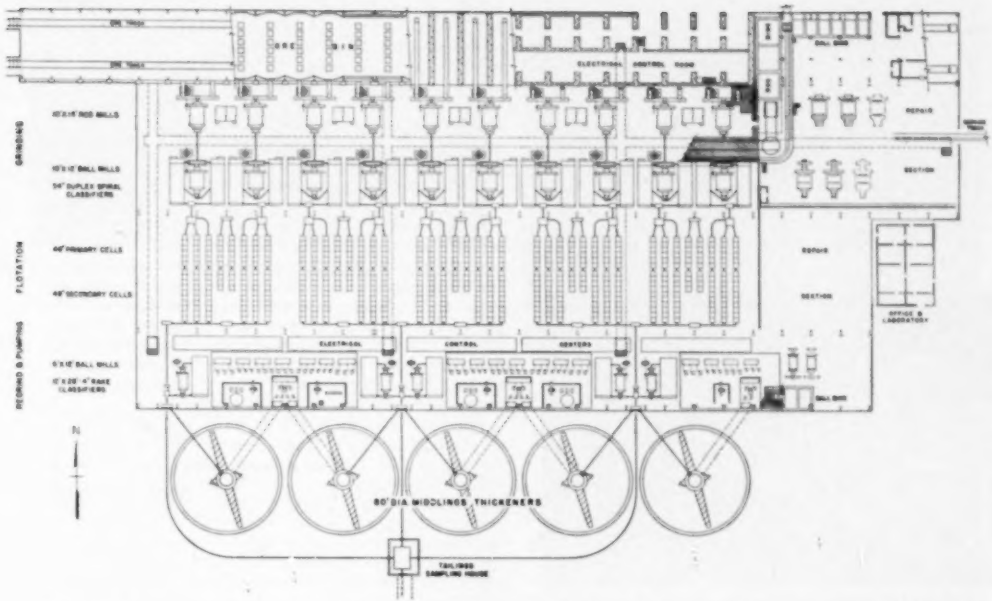
Two rows of six feeder openings per section reduced dead storage to a minimum and permitted the use of two separate horizontal 36-in. feeder conveyors running transversely across the bin directly into the grinding section. These feeder conveyors are located on the centerlines of two adjacent 22 ft 6 in. bays and each discharges onto a short 24-in. automatic weighing cross conveyor which carries the ore directly to the feed chute ahead of the rod mill. This arrangement provides two complete independent feed conveyor units each capable of handling the entire amount of nominal feed per section with ample excess capacity. The weighing conveyors automatically control the rate of feed to the rod mill by varying the speed of the preceding feeder conveyors.

Grinding Circuit

On the basis of the successful grinding tests conducted at Anaconda, with 9x12-ft rod mills in series with Hardinge ball mills, it was decided that two-stage grinding would be used at Chuquicamata. It was further decided that a 10x14-ft rod mill operating in open circuit, in combination with a 10x12-ft overflow type ball mill, operating in closed circuit with two 54-in. duplex spiral classifiers, would provide the most evenly balanced operation.

Normally, feed will be drawn from only one feed opening for each feed conveyor at any one time to minimize spillage and possible belt tension. As a result, horizontal rack and pinion gates have not been designed for feed regulation but rather to operate either fully opened or closed. As the ore enters the rod mill through the feed chute, water and lime are introduced through a funnel arrangement that can be easily observed by the operator. Upon passing through the rod mill the product spills into a specially designed housing which encloses the discharge end of the mill. This housing serves mainly as a support for the heavy plug assembly required to contain the rods within the mill proper. In addition, it serves to eliminate excessive splashing and

Plan of Concentrator



by means of removable perforated plate screens broken rods are kept from passing into the circuit. The next step in the flow of the material involves the laundering of the rod mill product directly to the ball mill scoop box where it combines with the classifier sands. During the design of the grinding section, it was anticipated that subsequent operation may prove it advantageous to launder the rod mill product directly to each of the classifiers. For that reason an adequate drop between the rod and ball mills has been provided making it possible to launder directly from the rod mill to the classifiers at a slope of 1¾ in. per ft. The classifiers are arranged with one duplex on each side of a ball mill. This design made possible the shortest crane span over the grinding section and an overall minimum drop through the section. A 78-in. radius combination double scoop and drum feeder designed to handle a 400 pct circulating load as well as the primary feed from the rod mill is used on the ball mill to close the circuit between the ball mill and the classifiers. The mill discharge is accurately divided by the use of a specially designed splitter assuring an equalized load on each classifier.

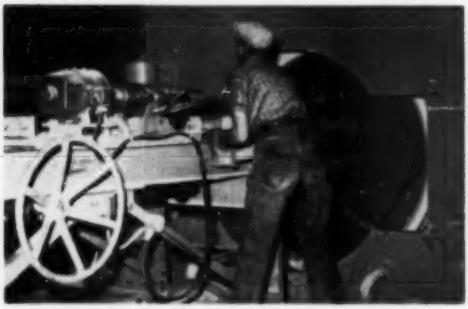
An added feature of flexibility in the grinding section involves the mill drives. Both rod and ball mills are driven by identical 800-hp synchronous motors. Although the operating speeds of both types of mills are different, identical 19-tooth pinion assemblies are used. However, the operating speeds of both the rod and ball mills can be changed by substituting 18-tooth pinions for less speed and 20 or 21-tooth pinions if greater speed is desired. The pinion housings and mill sole plates have been provided to accommodate any of the sizes mentioned. In addition to the possible change in the speeds of the mills, the spur pinions for the classifier drives can be changed to vary the speed up to 20 pct faster or slower.

When relining or major repair work is required on either the ball or rod mills they can be picked up as a unit with their entire load including the complete trunnion bearing assemblies by a 200-ton overhead crane, taken to the repair bay, and replaced with a spare mill. This crane is also equipped with a 25-ton auxiliary hook for handling the mill motors and other lighter loads.

Rods and Balls

The repair section is located at the east end of the grinding and ore bin sections and occupies with all its various facilities approximately 24,000 sq ft of floor space. Three repair stands are provided for each type of mill. These stands, made up of greased steel cradles on a pair of concrete piers, have been designed to support a completely assembled mill by its shell. With the mills supported in this manner it is comparatively easy to do any type of work on them using the 25-ton repair crane and small industrial boom trucks. Ample storage facilities for rods and balls have been included in the repair area at the east end of the ore bin section along the north wall. Balls are delivered and unloaded directly to the ball bins from an adjacent track running along the entire north side of the building. Additional storage capacity is available by making use of two ball bins located at the extreme northeast corner of the building. These bins have been installed at this time to facilitate construction for any expansion program. A magnet attached to a hand controlled electric hoist operating on a mono-rail directly above the ball bins is used to load a charge hopper which in turn empties an 8 hr ball charge for a single mill into a small car. A small battery operated truck hauls this ball car on a narrow gage track running the entire length of the grinding section. Loading chutes located at the centerline of each ball mill feed the balls directly to the mill

through the drum feeder. Rods, also delivered to the mill in railroad cars along the same track, are transferred to storage bins by means of an overhead cab-controlled electric hoist which is also used for removing them from the bins and placing them on a specially designed rod charging machine. In addition to the inside rod storage, additional facilities have been provided outside the building, served by the same overhead hoist. A battery operated rod charging machine, specially designed, travels on a wide gage track alongside of the ball track directly in front of the line of rod mill discharge boxes. This car can handle rods up to 4 in. diam and is capable of charging rods at the rate of one every 5 sec.



This specially designed rod charging machine travels on a wide gage track alongside the ball track directly in front of the line of rod mill discharge boxes.

The storage bins for both rods and balls are sand bottomed and lined with plate for protection. Accurate ball and rod charging records are kept by automatic recording track scales located on both charging tracks. Battery chargers are conveniently located in the repair section adjacent to the ball and rod charging tracks thus making it possible to charge batteries without removing them from their respective units.

The overflow from the two classifiers in each grinding section is collected in a common junction box and then laundered directly to the flotation section, where it is sampled as it enters the rougher flotation distribution box.

Rougher Flotation

The rougher flotation section for each grinding unit is made up of three machines of 3 two-cell primary roughers and 5 two-cell secondary roughers. Only one cleaner machine consisting of 3 two-cell primaries and 2 two-cell secondaries is provided for each of the grinding sections. The flotation cells are all Agitair machines using mechanical and air agitation. Mechanical agitators are V-belt driven by one vertically mounted motor for each two-cell unit. Air agitation is provided by a blower system supplying air at a pressure up to 1¼ psi and as much as 50 cfm volume per cell. One blower for each two sections is located in the pump station directly south of the respective sections they are intended to serve. The discharge lines from the blowers feed directly to a continuous header located beneath the platform at the tailing end of the flotation cells. This arrangement permits a more uniform pressure to be maintained during operation. Individual valves control the air at the headers to each machine.

Sufficient space has been provided in the flotation section to install an additional complete rougher machine and a comparable increase in cleaner capacity if the grinding section proves to have higher capacity than anticipated. Another feature is the arrangement of the flotation machines parallel to the axis of the complete section. This scheme has several advantages. First, all the distribution boxes are arranged in a single line at the same elevation and are served by the same continuous platform. Then, all the machines are equally spaced and at the same elevation with exception of the secondary cells which are 8 in. below the primary cells. This overall arrangement required less headroom for the installation and operation of a 5-ton crane spanning the 88-ft wide section. In addition, less drop in the launder collection system resulted because of the general flow of the feed and flotation products towards their ultimate destination. Adequate headroom has been provided below steel framing supporting the cells and platforms to facilitate cleaning and general launder maintenance.

All the flotation products with the exception of the tailing are laundered directly to pump sumps located in the section directly below the flotation section. At this stage in the flowsheet, it was found more convenient to deviate from the single independent section idea and combine the treatment of the middlings from two sections rather than one. In this way spare pumping facilities were kept to a minimum, and more economical sizes of regrind and middlings equipment were made possible.

Middling is collected from the primary roughers and after being laundered to the pump sump, where it is sampled, it is pumped to the middling thickener distribution box.

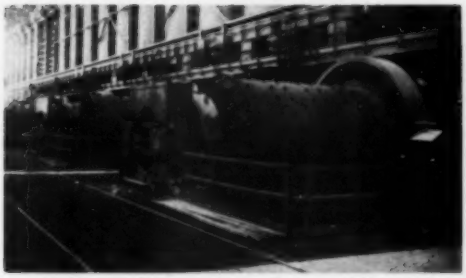
Middling Treatment

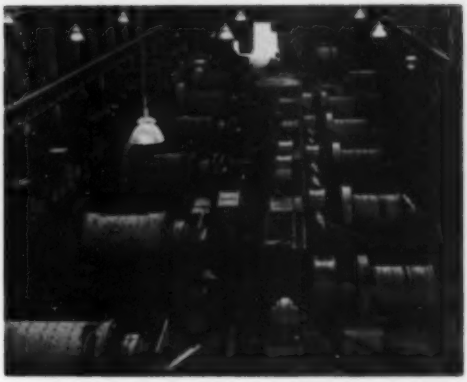
The middling tanks are 80-ft diam torque-type thickeners and although one thickener tank has been provided for two mill sections, added flexibility has been made possible by using a distribution box for adjacent pairs of thickener tanks. In this way, any of four adjacent mill sections can operate with either of the two tanks serving that group.

The thickened middling is piped in tunnels that extend from directly under the center of the tank back to the respective pump pit for the section it serves. This new feature avoids the need of burying any section of the pipe in concrete and thereby makes possible access to all the underflow piping for removal and inspection. The percentage of solids in the underflow is controlled by various sized orifices that are readily installed at the discharge end of the pipe.

With the middling tanks located in the yard immediately south of the concentrator, the pumps handling the underflow, are about thirty-two ft below yard grade. However, this relative depth was minimized to some extent by the added feature of having the spigot pits located within the concentrator pump pits which in themselves are about ten ft below yard level. In addition, the spigot pit is entirely open at the surface making it accessible by the overhead crane serving that section.

The thickened middling is pumped to a junction box at the regrind section, where the pulp then flows to the regrind distribution box where it is sampled. As in the case of the middling tanks, each regrind unit handles the product of two primary mill sections. Thus it is possible to attain the same degree





A combination of 10x14-ft rod mills, operating in open circuit, and 10x12-ft overflow-type ball mills, in closed circuit with two 54-in. duplex spiral classifiers, provided the most evenly balanced operating arrangement. Above is closeup of the rod mills and the grinding bay is seen at right.

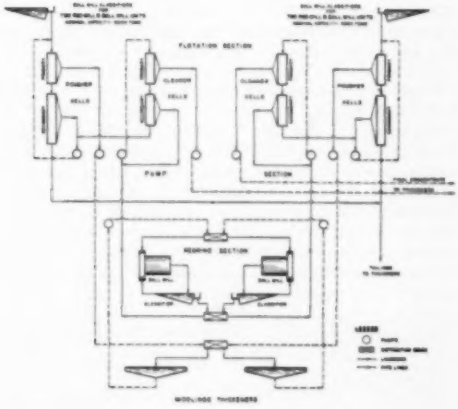
of flexibility as that previously described for the middling thickeners.

Regrind Circuit

Each regrind circuit consists of a 6x12-ft ball mill in closed circuit with a 12x28-ft type QSF Dorr classifier. Scott sandwheels, 6 ft diam x 14 in. wide, are attached to the discharge flange of the mills to aid in closing the circuit without the need of classifier excessive slope.

The entire regrind and pump section is served by a 75-ton crane having two 37½ ton sister hooks for removing and replacing mills and a 10-ton auxiliary hook for handling pumps, motors and small loads.

Flowsheet of Regrind Circuit



The repair area is located in the extreme east end of the regrind and pumping section where two repair stands similar to those used for the larger mills have been provided along with a spare mill.

Balls are delivered and fed to the regrind mills in the same manner as those for the larger mills. The ball track extends the full length of the section from two 300-ton storage bins provided along the south wall in the repair section.

The regrind circuits have been located with sufficient elevation to permit the classifier overflow to be sampled and then laundered to either of the two adjacent pump pits serving their respective circuits. With this arrangement it is actually possible to isolate any given regrind circuit while making use of the one adjacent to it.

After collecting the regrind classifier overflow it is pumped to a distribution box in the flotation section ahead of the two cleaner machines included in the grouping of the two sections served by the single regrind circuit.

All the recirculating pumps are also located in the pump pit. One group of these pumps is used in pumping the combined secondary cleaner tailing and secondary rougher concentrate back to the rougher flotation distribution box after each has been sampled. Another group recirculates the secondary cleaner concentrate back through the cleaner machines after it too has been sampled.

The final concentrate is collected from the primary cleaners and pumped directly to the concentrate thickener distribution box. The total number of pipe lines to the thickeners have been reduced by collecting the concentrate in only two sumps in the pump pit. However, the product from each of the sections is sampled before collection.

Sampling

Special attention was given to sampling operation. A major portion of this work is done in the pump pits where automatic samplers have been installed in a line over the single row of pump sumps. In this way it was possible to reduce the total number of machines by installing two or three cutters on specially designed carts operated by the sampler mechanism over each of the sumps. In addition, this arrangement makes observation and maintenance an easy matter from a continuous platform set directly on the pump sump.

Vezin-type sample splitters operating simultaneously with the samplers have been located behind the pump sumps directly over a common trough where the excess sample is collected and combined with the floor drain system.

Floor drainage is handled by a system of concrete launders located along the south wall of the grinding section and the north wall of the pump pits as well

as floor drains in the mill repair, regrind, and pump sections. As a result, spillage and floor waste as well as mill contents from the repair sections ultimately reach service sumps located in each of the middling spigot pits. Float valve activated pumps then pipe this material to the middling distribution box for further milling. An emergency overflow pipe arrangement has been installed connecting the grinding section floor drain launder to the tailing system in the event of power failure and possible delay in closing mill feed water lines.

Tailing Thickeners

One of the prime requisites in mill design is the collection and laundering of the final tailing to the thickener tanks by gravity. The importance of this fact becomes more apparent when it is realized that about 95 pct of the original feed is rejected as tailing.

Final tailing is collected from the secondary rougher flotation discharge boxes. Rubber lined pipe elbows direct the tailing into the collecting launders located just below the cell platforms. These launders carry the tailing to sampling boxes where two adjacent sections are sampled separately. From this point the tailing is laundered to a junction box where it combines with the tailings of two adjoining sections. The launders from these junction boxes run directly across the regrind and pumping section between each pair of regrind mills just above the mill floor then out into the yard between the middling tanks to the tailing sampling house. This arrangement makes possible the handling of tailing through the mill in easily accessible launders that are all above the mill floor rather than in deep trenches or tunnels as in many previously built concentrators.

Final samples of tailing are taken at the sampling house before the combined tailing is carried in a single concrete launder through a tunnel under the main yard directly to the tailing distribution box located on the slope between the main yard and the thickener tanks.

The tailing dewatering system consists of four traction type 300-ft diam thickeners which ultimately will handle tailing from 12 grinding units. The tanks are arranged in pairs on two different yard levels. Those closest to the distribution box are about 30 ft below the main yard and approximately 20 ft above the low pair. This arrangement conformed suitably with the natural slope of the ground, keeping a greater proportion of each tank on firm footing and reducing the number of piers required on those portions of the tank walls that rested on fill material. No unnecessary overall loss of height was experienced in the complete operation of tailing distribution and underflow collection.

The tailing spigot pit was located between the pair of tanks on the lower yard to reduce the overall depth. The tunnels from the pit to each of the tanks on the lower yard were extended completely through the center piers and continued on under the center piers of the upper level tanks in the same manner as those for the middlings thickeners. With each tunnel accommodating the underflow lines from two tanks the total length of tunneling required has been reduced and maintenance and control made correspondingly easier. A small automatic passenger and freight elevator has been installed in the spigot house for quicker and easier access to and from the orifice control station some 40 ft underground.

A 750,000 gal concrete return water tank is located on the lower tank yard near the base of the slope from the upper yard. This tank receives the overflow directly from each of the tailing thickeners for return to the head tanks. To consolidate pumping facilities, the collected overflow from both the middlings and concentrate thickeners are also piped by gravity to this tank.

Vertical deep-well type return water pumps are housed in a structure built directly on top of the overflow tank. They are controlled automatically by the water level and can be easily serviced by an overhead crane. In the event of power failure, an overflow pipe has been installed to direct excess flow to the tailing spigot house where it would combine with the tailing underflow and be laundered to the disposal area.

Concentrate Dewatering

The pipe lines carrying the concentrate from the concentrator to the thickener tanks run completely exposed above ground for a maximum length of about 2000 ft. The distribution box into which these pipes discharge is centrally located between a cluster of three tanks directly west of the filter plant. These tanks are located on the main yard at the same elevation as the middling thickeners. They are 130-ft diam torque-type thickeners having central and peripheral froth baffles to prevent froth interference at the overflow.

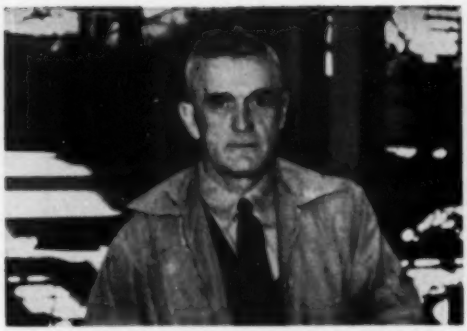
The concentrate underflow pipes are carried in tunnels from under each tank to a common spigot pit located within the Filter Plant. From this point the thickened concentrates are pumped to an intermediate distributing box which has been elevated sufficiently to permit the pulp to be by-passed and laundered back to the distribution box at the thickeners if the occasion should demand. However, during normal operation the concentrates are sampled and go to a distributing box ahead of the filters.

One notable feature of this by-pass design is that the entire arrangement avoids the use of any valves in the pipe lines.

Disk-type filters are used in the filter plant. These machines are 12 ft 6 in. diam having seven leaves each. Although only three filters are required for normal operation of the initial plant, an additional machine with its necessary auxiliaries has been installed to maintain production during periods of necessary repair. An overhead crane manually operated with an electric hoist is used to replace disk segments as well as for other maintenance work. The room in which the filters are located is completely partitioned from all other mechanical and electrical equipment as a protection against dust.

The filters are all arranged in a single row with each machine discharging directly onto a separate belt running at a speed synchronized with that of the filter. This scheme insures a uniform discharge of filter cake on to a collecting belt, avoiding spillage and any extreme variation of belt loading which may cause inaccurate weighing.

After the collection and weighing of the concentrates at the filter plant, they are conveyed to a junction house from which they are carried by conveyor either to the smelter or to the bedding system for storage.



D. S. (Scotty) Sanders, concentrator superintendent, is heir to all the usual problems of a new plant. He brings a wealth of experience to the job, with a history of experience which includes Cerro de Pasco.

E. W. Witcomb, assistant concentrator superintendent, is in the way of founding a family tradition at Chuqui. His father worked there, and now, Ted has come up through the ranks to his present position.

Chuquicamata Sulphide Plant:

Concentrator Operation

D. S. Sanders and E. W. Witcomb

Superintendent, and Asst. Superintendent of Concentrator, Chile Exploration Co.

FLOTATION tests, from which the flowsheet for the concentrator was determined, were made in a pilot mill at Chuquicamata and later comparative testing of full size flotation cells was done at the Andes Copper Mining Co. concentrator in Potrerillos. The principal sulphide copper mineral is chalcocite with varying amounts of covellite, enargite, and chalcopyrite. The oxidized portion is principally antlerite, a green basic copper sulphate. The gangue minerals include quartz, sericite, pyrite, feldspars and kaolin. Chuquicamata pit ores have proved to be hard grinding. The results of the flotation and metallurgical work prove that good extraction and concentrate grade can be obtained under a rather wide range of conditions.

Metallurgy

In general the metallurgy of the ore requires a fairly close control of the grind to the primary rougher flotation and of the regrind of the primary rougher concentrate before cleaning. Regrinding is necessary to reduce the insoluble content of the final concentrate since a large part of the middling combinations are between the copper and silica minerals. The alkalinity of the pulps can vary between wide limits and control of the percent solids gives the desired grind.

Alkalinity Control

Arroyo Salado water is supplied to the concentrator head tanks. It contains about 5000 ppm of salts but test work shows no deleterious effect. Actually this water permits a decrease of as much as 1 lb of lime per ton of ore as compared to San Pedro water, which is used for leaching oxide ore. About one quarter of the water flow through the concentrator will be new water from the reservoir. The remainder will be reclaimed from the thickeners. For primary grinding and rougher flotation, new and reclaimed water are combined in the head tanks. To avoid erratic results, only new water is used in the regrind system.

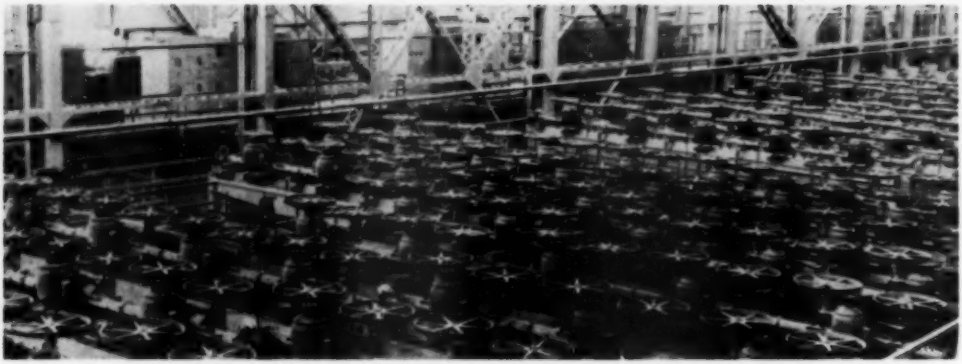
Milk of lime is added to the rod mill feed chute to obtain the maximum benefit of mixing with the ore. This protects the equipment from corrosion due to entrained acids and soluble copper salts. Lime consumption is 3 to 5 lb per ton of ore.

Grinding and Flotation

The six bin gates over each belt feeder are opened in controlled combination and sequence to keep a live bed of ore in the bin to prevent arching and to obtain mixing. The milk of lime added ahead of the rod mill is regulated to suit rougher flotation requirements.

The primary promoter reagent, Aerofloat 238, is added to the 10-mesh discharge of the rod mill as it is laundered to the ball mill scoop box.

The classifier overflow is regulated to produce a grind of about 12 to 15 pct +65 mesh and 60 pct −200 mesh. A secondary promoter Z-4 (aml zanthate) and the pine oil frother are added in the overflow launder where a pH of 10 to 11 is maintained. Comparatively heavy beds of froth on the



Comparatively heavy beds of froth on the rougher cells minimize the adjustments of air and pulp level.

rougher cells minimize the adjustments of air and pulp levels.

The froth from the primary rougher cells of two sections is combined and thickened for feed to the regrind circuit. New water, and milk of lime for required alkalinity, are added ahead of the ball mill. The classifier overflow at an approximate 200-mesh grind and 9.0 to 9.5 pH is pumped to cleaner flotation, and is split equally to two cleaner sections.

The final concentrate, after thickening and filtering, is ordinarily bedded for furnace charge, but it can be conveyed direct to the smelter with flux added on top of the concentrate as it passes through the junction house.

Tailing and Return Water

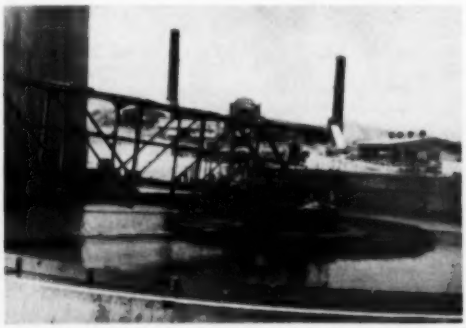
The final flotation tailing is thickened to 45 or 50 pct solids controlled by orifice size in the underflow lines. This density permits the maximum return of water while maintaining the proportion of solids desirable for flow in the tailing ditch and pipe. All thickener overflow and other reclaimable water flows to the return water tank in the tailing thickener area for pumping back to the head tanks. The pump operator, guided by remote indicators of head tank levels, has charge of tailing thickener operation and pumping. Another operator regulates the thickener underflow, using density scales for pulp control and measuring devices for clear water depth.

Metallurgy is comparatively simple. Concentrate grade is easily regulated by control of the iron floated. A concentrate of 40 pct is now produced which in turn makes a reverberatory matte containing 44 to 46 pct copper. The concentrate is practically self-fluxing. Lime discard or undersize from the lime burning plant will be used as a basic flux.

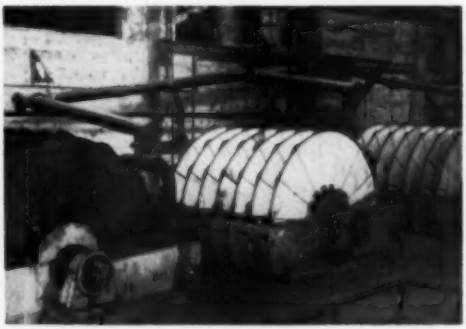
The flowsheet provides for the re-grinding of the primary rougher concentrate before cleaning, so that the copper minerals therein are liberated from the gangue to make a final copper concentrate of the proper grade and with the least amount of insolubles.

Since the main copper mineral is chalcocite, the problem consists of recovering sufficient pyrite in the final concentrate to provide the smelter with the desired iron and sulphur. Fortunately there is sufficient recoverable pyrite in the ore to permit close control of the concentrate pyrite content.

By the use of small amounts of Z-4 in the rougher and cleaner circuits, a close control of pyrite has been found possible in the final concentrate. Aerofloat 238 has proved to be a good promoter for chalcocite but not for pyrite. The flotation pulp is alkaline and the presence of lime inhibits the flotability of pyrite so that some actual pyrite promoter was found necessary to float more pyrite in the concentrate. By simply replacing a part of the 238 with Z-4, until the desired pyrite content of the concentrate is reached, it has been found possible to obtain concentrate of suitable composition.



One 80-ft middling thickener is provided for two mill sections; flexibility is achieved as any of four adjacent mill sections can operate with either tank serving that group.



Four 12 ft 6 in. diam, 7-leaf disk filters are housed in the filter plant; only three machines are needed for normal operation, the fourth being for stand-by.

Chuquicamata Sulphide Plant:

Tailing Disposal

R. M. Kuralt
Design Engineer,
Anaconda Copper Mining Co.

A natural basin flanked by low-lying hills and aided by evaporation provides ample tailing storage and excess water disposal.

CONCESSION from the Chilean government granting the company use of the Rio Salado water stipulates that a minimum of 35,000 metric tons of such salty water must be diverted from the Salado daily, and be disposed of in a manner not to contaminate the Rio Loa to the east of its confluence with the Rio San Salvador at Chacance, Fig. 1. Consequently, the selection of a suitable site for the sulphide plant tailing disposal had the additional problem of the disposal of excess brackish waters.

Studies of the terrain in the area directly to the south of the plant indicated that construction of tailing ponds would be costly, and generally unsatisfactory due to excessive slope of the ground, lack of flanking hills, danger of raising the groundwater level thus causing springs along the Antofagasta-Bolivia railroad; and the possibility of creating a dust nuisance at the plant site or at Calama due to blowing sands. Further to the southwest the San Salvador river canyon offered some possibilities and surveys were made at sites A to D, inclusive, Fig. 1. Here again costly dams would have been necessary to impound the tremendous quantity of tailing produced by the new plant, and provision would have to be made to handle a considerable flow of water through the disposal area to the west of the Rio Opache. Additional surveys were made in the area to the southeast of the plant which brought to light the existence of a natural basin of about 14½ square miles, flanked by low-lying hills, and requiring only the most rudimentary dams to close the outlets at the western end. It has been estimated that the Salar de Talabre, as this basin is called, has the ultimate storage capacity of 1 billion tons of tailing with dam heights of only 60 ft.

Pipe Line Layout

After the decision had been reached to use the Salar de Talabre for the main disposal area various methods were considered for transporting the tailing from the 300 ft diam thickeners to the disposal site, 17.7 miles away. The first plan was to construct a concrete launder on a 1½ pct grade but a survey of this grade line indicated it would not lead directly into the Salar but would terminate somewhat to the north, and from that point on the slope would be considerably less than 1 pct along a natural watercourse. A slope of 2 pct or even 2¼ pct in such a rough watercourse or ditch is the ideal, judging from the success of operations at Cananea. Since a simple ditch is the most economical vehicle for the transportation of tailing, it was decided to use one with a 2¼ pct grade for as much of the distance as was possible with the existing terrain and to construct a concrete headbox, or forebay, at the terminus of the ditch to supply a steel pipe line leading the rest of the way into the Salar.

Referring to the maps of the Chuquicamata area outlining the tailing disposal system, Figs. 1 and 2, one can trace the flow of the tailing from the thickeners to the Salar as follows: The 8-in. spigot pipes from the thickeners (1) discharge into a concrete sump feeding a concrete launder with a 2 pct slope. The launder carries the tailing into a concrete junction box (2) located south of the sulphide plant. The purpose of the junction box is to provide emergency facilities for diverting the tailing flow from the main disposal area to the emergency area. It also provides a junction point for the launder from the sulphide plant's future east extension. The box is fitted with a pair of radial gates controlling flow into the main tailing ditch (6) or the emergency ditch (3) which terminates on the pampa. All ditches are of the same cross section, namely 4 ft wide at the bottom with 1 : 1 side slopes and with a minimum depth of 2½ ft. Where the nature of the ground demands it, the ditches are lined with stone rubble. The overflow from the plant's 10 million gal reservoir (4) is led into the main tailing ditch by way of a spillway ditch (5). The main ditch terminates in a trash-screen fitted, concrete forebay (7) which serves as a headbox for a 24 in. O.D. x ⅜ in. wall, Dresser coupled, steel pipe (10) and a 12 in. x 8-gage spiral welded, wedge-lock coupled, steel pipe (8). The forebay is fitted with two compartments, one for each pipe, and flow into each is controlled by radial gates. Each compartment is fitted with a 6-in. drain pipe controlled by a rubber pinch valve. To protect the forebay from the abrasive effects of the tailing, the sides of the compartments will be lined with old

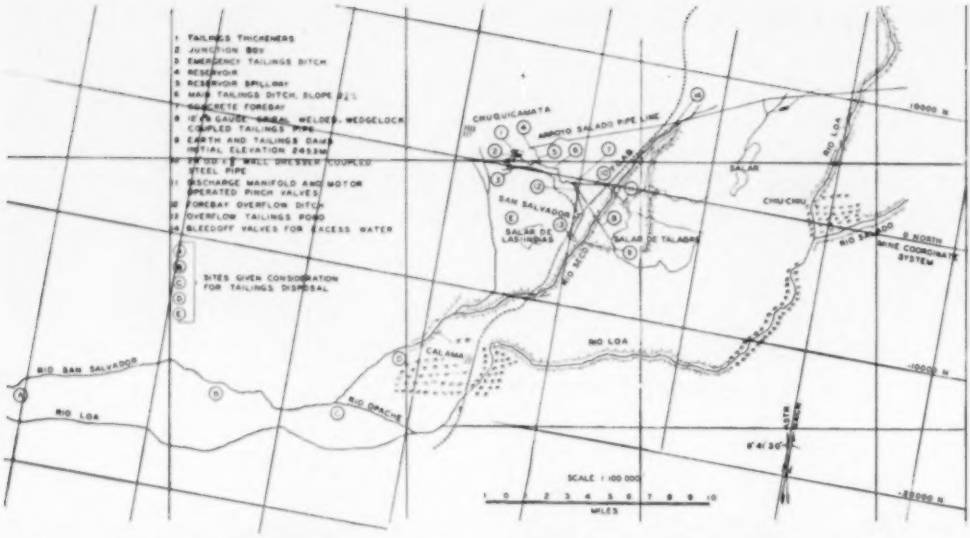


Fig. 1—Plan of area around Chuquicamata shows rivers and possible tailing disposal sites. Terrain south of the plant, site E, was ruled out by costly construction, danger of ground water level, and possible dust nuisance. Sites A to D required costly dams for tremendous quantity of tailing handled. Discovery of the Salar de Talabre southeast of the plant, provided 14½ sq mile basin flanked by low-lying hills and needing only rudimentary dams.

conveyor belting, and a wear bed of large stones will be placed at the bottom. The forebay is also fitted with an overflow weir which empties into a ditch leading to an overflow pond formed by low dikes (13). If the tailing pipes choke up, the overflow pond will take care of forebay overflows until the flow can be diverted to the emergency area south of the plant (3).

Dam Construction

The 12-in. pipe line is in the form of an inverted siphon with a low point where it crosses the Rio Seco. It runs from the forebay directly to the three dam sites at the western end of the Salar. Prior to the start of operations earth dams were constructed at two of these points to an elevation of 2453 meters. At dam site A, Fig. 2, the maximum height of the starting dam was only 4.26 ft; at dam site B no preliminary work was required; and at dam site C the maximum height was 17.39 ft. Since test drillings at these localities indicated that the ground contained soluble salts that would be unsuited for dam building, it was considered necessary to face the embankments with an impervious blanket of fine tailing materials. To do this the 12-in. pipe is led along the berms of the embankments, and 1½ in. branch connections controlled by rubber pinch valves are fitted to the line at short intervals. These pipes are fitted with rubber hose sections for flexibility when directing tailing to any section of the dam face, or when raising the dam crests in the future. By using small sized branch lines, the eroding effects of a large stream of tailing is avoided, with the consequent elimination of the constant need to rake up the slimes on the faces of the dams. The end of the line, at elevation 2453 meters, is fitted with a 12-in. pinch valve which can discharge the full flow of the 12-in. line into the Salar when so desired. The hydraulic gradient of this line is 1.26 pct for the starting up operations. A drain valve is fitted at the low point of the inverted siphon. The main purposes of the 12-in. line are to transport tailing to the dam sites for use in construction operations, and to serve as a starting up line until sufficient quantities are available to operate the 24-in. main pipe line. Until field operations prove otherwise, the designers consider it wise to limit the pipe line velocities to the range of 3.5 to 6.5 fps. This sets the tentative capacity of the 12-in. line at 1230 to 2290 gpm and of the 24-in. line at 4630 to 8600 gpm, corresponding approximately to concentrator operations of 2 to 3 and 6 to 12 grinding units respectively. When operations of the concentrator supply less than the minimum volumes, either the emergency area south of the plant or the forebay overflow pond can be used temporarily.

The main 24-in. tailing pipe line is laid downgrade all the way with no low point requiring drainage in case the line is shut down. It terminates at the rim of the Salar with an elevation of 2485 meters, giving a hydraulic gradient from the forebay of 1.09 pct. Two 24-in. discharge manifolds are fitted at the end of the pipe line with geared plug valves controlling the flow into each. Three 12-in. branch outlets are fitted on each manifold with rubber pinch valves controlling the flow from each branch. Two pinch valves on one manifold are motor operated, and controlled by float switches actuated by the pulp level in the forebay. In this manner the motor-operated discharge valves will be automatically closed or opened in sequence, as the pulp level in the forebay falls or rises, due to fluctuations of flow from the plant. This will prevent the pipe line from operating in a partially full condition with consequent scouring due to high velocities on the steeper grades, and will tend to eliminate forebay overflows. A high and low level alarm is also provided to warn

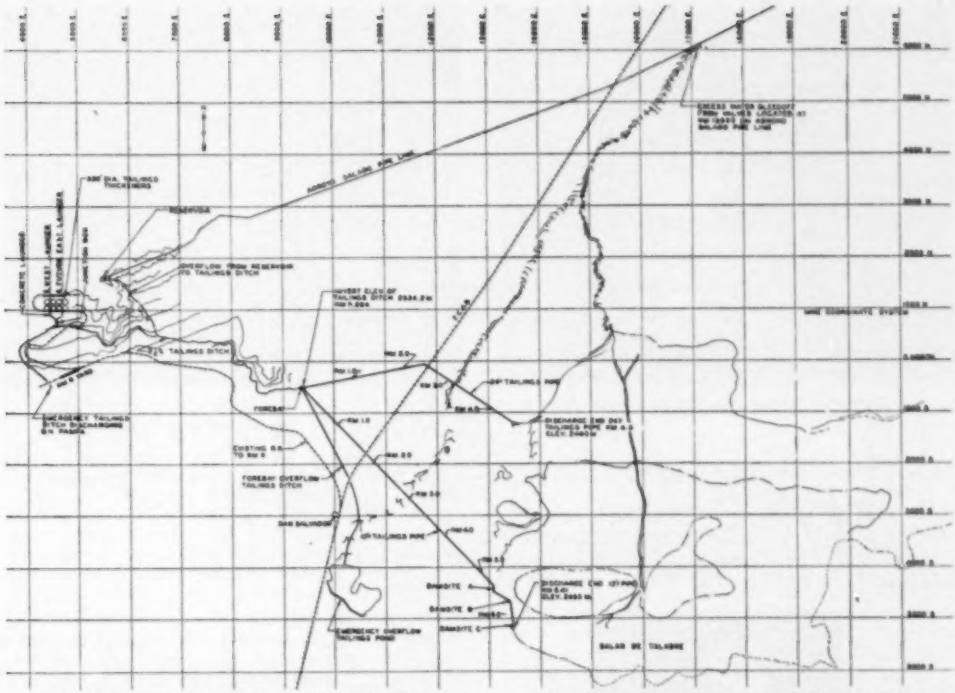


Fig. 2—Flow of tailing to the Salar de Talabre makes maximum use of natural grade. Problem of insufficient grade for a simple ditch all the way was solved by combination of ditch and steel pipe. Ditch at 2¼ pct gradient terminates in concrete headbox, which supplies steel pipe leading to the Salar at reduced gradient. Tailing may be diverted to an emergency area near the plant, and at headbox there is provision for overflow diversion.

the operator at the discharge manifold should additional valving be required.

Evaporation Solves Contamination Problem

The tailing from the pipe outlets will run into the Salar and cut its own channels leading to the lower reaches of the basin where a large lake will form. No manual labor will be required at the outlet of the main line for many years, except to operate valves and to control erosion. The large surface area of the lake formed will evaporate water at a rate estimated to be as high as 1 cm per day. No attempt will be made to reclaim water from the pond for further plant use. The overflow from the plant reservoir will be kept to a minimum by bleeding off all excess water from various sized valves located at station 13.55 Km on the Arroyo Salado pipe line (14) in order to maintain a pulp density of 50 pct solids by weight in the pipe lines, thereby avoiding varying critical velocities. Such water as is bled off will find its way into the tailing pond to evaporate. As plant operations go forward on a larger scale, the need for wasting water to the tailing pond will disappear; then Chilean government's requirement of removing 35,000 metric tons of water from the Rio Salado daily will be met by the plant's increased demand for water. It will only recur in the event of a plant shut down, or reduction in operations.

Vertical deep-well pumps housed on top of 750,000 gal concrete thickener overflow tank return water to concentrator reservoir.

All railroad crossings for ditches and pipe lines are through 30 in. I.D. x ½ in. wall corrugated steel culverts, except the emergency ditch, which crosses under the railroad west of the Km 6 yard in a reinforced concrete culvert.

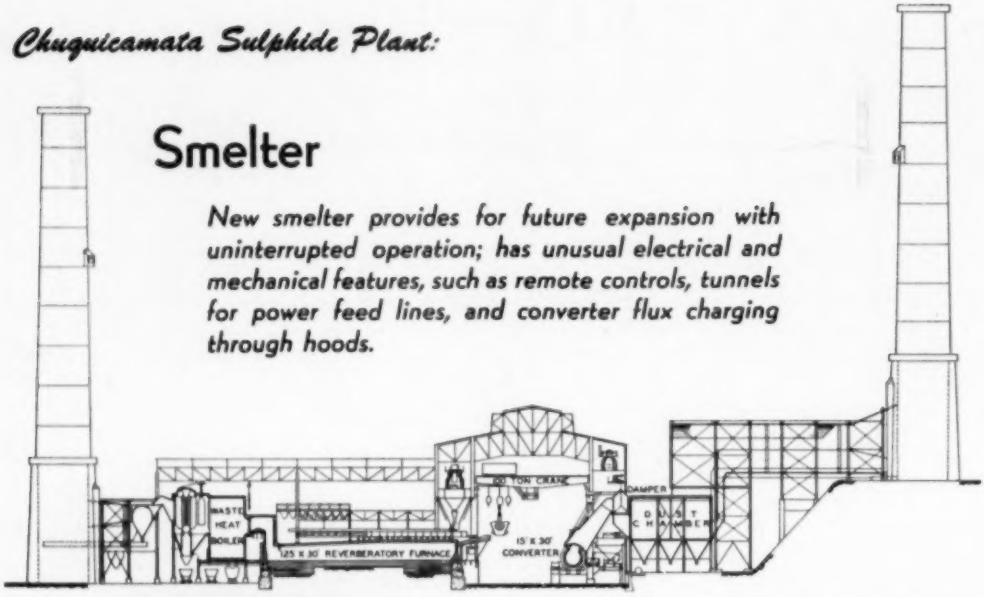
The pipe lines are supported and anchored by terrepleins similar to those used on the Arroyo Salado pipe line.

When future plant expansion requires more tailing disposal capacity, the 12-in. pipe will be replaced with another 24-in. pipe. The combined capacity of two 24-in. lines will be adequate for a concentrator having a capacity of 60,000 tons.

Chuquicamata Sulphide Plant:

Smelter

New smelter provides for future expansion with uninterrupted operation; has unusual electrical and mechanical features, such as remote controls, tunnels for power feed lines, and converter flux charging through hoods.



H. G. Dwyer

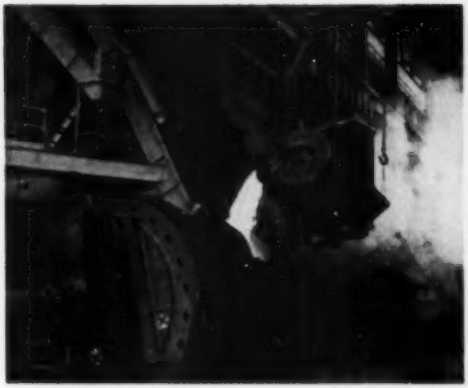
Design Engineer,
Anaconda Copper Mining Co.

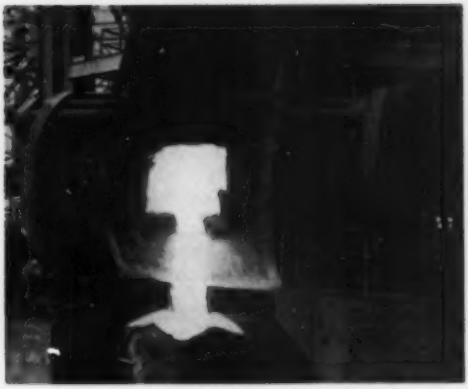
CONSIDERATION for future expansion influenced the design of the new smelter at Chuquicamata. The section of the smelter now going into operation, while large, represents only little more than half of the ultimate plant. Equipment for the direct smelting of raw concentrates has been arranged in the conventional manner although needless details have been eliminated wherever possible while mechanical and electrical features not usually found in copper smelters have been incorporated.

Access is afforded to all equipment but platforms and walkways have been held to a minimum. The poling and other platforms in the converter aisle, normally subjected to battering by ladles and other heavy equipment in transit, are of concrete construction with steel grid armor embedded in the exposed surfaces. Noticeably absent are the many platforms formerly required for the handling of converter flux since this material is charged to the converters through the side walls of the converter uptake hoods. Clear space with ample headroom is provided under each converter and converter puncher's platform to permit clean-up by small bulldozers. Remote controls have been introduced at many points. Electrical control rooms and control centers have been conveniently located. An electrical tunnel with cross tunnels and escapeways is situated under the entire length of the converter aisle for power feed lines.

Skull breaking facilities are in a separate building near the smelter. A silica-slurry building with brick storage area is adjacent to the smelter building so that elevator service is available to both buildings. A smelter office, a change house and lunchroom are provided in convenient locations. Units of the smelter as presently constructed are shown in the plan and table on page 1194.

The ultimate smelter will house 7 reverberatory furnaces, 14 waste-heat boilers, 9 Peirce Smith converters, 4 casting furnaces serving 2 casting wheels, and one holding furnace serving a straightline casting machine. Additional crane requirements in the converter aisle will necessitate the installation of special hoisting equipment in space already provided in a raised section of the roof over the casting end of the aisle. Thus a complete crane may be lifted high enough above the crane rails to permit the passing of another crane underneath thereby giving all cranes full use of repair facilities already installed. For the collection of gases, 2 dust chambers, each with separate stacks, will collect converter gases and 2 balloon flues with separate stacks will collect reverberatory furnace gases. Extensions necessary to complete the ultimate plant will be possible with little interruption to normal operations since the future converters and reverberatory furnaces will be added at the east end while additional casting facilities will be added at the west end of the present plant. The future units may be added singly or all at one time depending on requirements.





Matte is poured to converter (left) and after blow is skimmed, (right) as years of planning and months of construction culminate in first copper from new sulphide plant.

The converter aisle floor is 6 in. above the surrounding yard level and is unpaved at present. Poured slag may be used later for localized paving under and in front of the converters if required by the operating staff. The floor of the reverberatory plant is 15 ft above the converter aisle floor. The poling platform and the combined anode platform and copper docks are 11 ft and 3½ ft, respectively, above the converter aisle floor.

Smelter Crushing Plant

Limerock, converter flux, smelter reverts, and small amounts of other smelter materials are crushed in the smelter crushing plant.

The materials are delivered to the crushing plant in railroad cars and received in a 300-ton track bin. From this bin a 48-in. pan feeder discharges to a 42x40-in. jaw crusher. The crusher product passes over a 5x10-ft mechanically vibrated screen (1½-in. opening), the oversize going to a #5½-standard cone crusher. The combined screen undersize and cone crusher discharge to a #5½-short head crusher preceded by two 5x10 ft mechanically vibrated screens in parallel (¼-in. opening). The materials are transported by belt conveyors. If stoppage or failure at any point in the system causes material to pile up from preceding equipment, this equipment is stopped automatically through electrical interlocks. This protection has been installed on all conveyor systems throughout the smelter area.

The several materials handled are crushed to various sizes. By means of suitable by-pass chutes the materials may be drawn from the crushing circuit at the desired size. The limerock is crushed to −1½ in. +¼ in. for burning and screened to −¼ in. for fluxing of concentrate. Mine ore containing a good percentage of free quartz is used for converter flux. It is crushed to −1½ in. The smelter reverts are crushed to −¼ in.

Miscellaneous smelter materials, not requiring crushing, are brought in through the track bin and diverted ahead of the jaw crusher to by-pass the crushing circuit.

The crushed converter flux, reverts, and limerock for fluxing are stored separately in compartments of a 5200-ton storage bin. From there the materials are delivered to their destination by belt conveyors.

Bedding Plant

From the filter plant the concentrate is transported to the beds by a belt conveyor system, including traveling trippers and an automatic weighing machine. If desired, the concentrate may be sent directly to the reverberatory feed bins, by-passing the beds. A tripper spreads the concentrate, flux, and reverts in layers over the length of the bed. Weights of all materials going to the beds are ob-

George E. Morris, smelter superintendent, transferred from Anaconda's Potrerillos, Chile, operations to take over at Chuquicamata.

Assistant superintendent Frederick Towell thinks Chilean climate and terrain compare favorably with Miami, Ariz., where he came from.

tained by weighing machines. This permits accurate fluxing and charge calculations.

There are three beds 50x360 ft, each having a capacity of 11,000 tons.

After a bed is completed and properly fluxed a reclaiming machine excavates the charge for the reverberatory furnaces. There are two reclaimers and one transfer car by which the machines are moved between beds and to the repair bay. Each reclaimer has a capacity of approximately 200 tons per hr.

The reclaimer scrapes the material from the end of the pile and deposits it by means of a drag bucket conveyor on to a belt conveyor in a trench alongside of each bed. The machine has a reciprocating harrow set at an incline a little steeper than the angle of repose of the concentrate. As the reclaimer advances down the length of the pile the harrow removes a uniform cross section of the layers of the concentrate, flux, and reverts. This provides a uniform reverberatory feed from each bedded charge.

A belt conveyor system transports the furnace charge to the reverberatory plant. The charge is weighed by an automatic weighing machine.

Lime Burning and Hydrating Plant

The lime burning and hydrating plant is operated to supply milk of lime for the concentrator.

Crushed limerock, $-1\frac{1}{2}$ in. $+\frac{1}{4}$ in., is stored in a 940-ton bin from which it is conveyed to a 50-ton surge bin ahead of the lime kiln. The kiln, 7 ft 6 in. x 315 ft long, is fed by a weighing feeder. It is fired with Bunker C fuel oil and has a rated capacity of 125 tons of burnt lime per 24 hr.

A ball mill and classifier are used to hydrate the lime. Insoluble cores produced as sands from the classifier are discarded. The milk of lime is pumped to the concentrator.

Reverberatory Plant

A traveling tripper, operating in a conveyor gallery above the firing ends of the reverberatory furnaces, delivers raw concentrate mixed with fluxing materials to charge bins, each having a capacity of 350 tons. Two such bins together with two smaller bins for silica or concentrate are provided over each reverberatory furnace. The small bins, each with a capacity of 100 tons, are used for fettling at the firing end of the furnace when converter slag is being returned. The traveling tripper also fills these smaller bins. Pan feeders, 54 in. wide, transfer the charge from the large bins to drag chain conveyors located over the furnace arches near the side walls of the furnaces. A pan feeder together with an adjoining drag chain conveyor operates as a unit and the drives of each are interlocked electrically so that neither one may operate without the other. The pair of pan feeders and drag chain conveyors over any reverberatory furnace may be operated simultaneously or separately to suit the smelting requirements. The charging rate is adjustable and is determined by the depth of the material being carried on a pan feeder. Fettling pipes, equipped with slide gates, direct the charge from the drag chain conveyors to fettling holes located in the furnace arch and spaced on 3 ft 8 in. centers to about 60 ft out from the firing-end wall. The slide gates are opened or closed manually from walkways provided along each side of the reverberatory furnaces.

The reverberatory furnaces are of similar design to furnaces now operating in the International smelter at Miami, Ariz., and were designed there under the direction of P.D.I. Honeyman, general manager of that plant. The furnaces are 30 ft wide by 125 ft long at the outside brick lines and will smelt about 650 tons of charge each per day. They are spaced on 64 ft centers. The roof arch is of silica brick, 20 in. thick, horizontal for about 58 ft and sloping from there to the throat at the uptake end. The side walls are of silica brick lined with basic (chrome magnesite) brick in the bath zone and around the tapping and skimming ports. The side wall thickness varies from a minimum of 1 ft at the top to nearly 5 ft at the bottom. The roof arch and side walls are insulated. The bottom of the furnace is of poured slag with a minimum thickness of 8 ft at the ends and 4 ft in the intermediate zone. The top of the slag bottom is at the same elevation as the floor of the reverberatory plant. A heavy concrete foundation retains the slag bottom, supports the furnace walls and buckstays, and takes the bottom thrust of these latter members. Pilasters, 18 in. wide, hold the buckstays out from the face of the furnace walls. The buckstays are on 5 ft 6 in. centers taking horizontal tie rods in the conventional manner. A longitudinal tie rod extends along the full length of each side of the furnace.

Converter slag is returned to the reverberatory furnaces in large matte ladles, each holding nearly 50 tons, by means of the converter aisle cranes and it is poured into a launder which enters the furnace through the center of the firing-end wall.

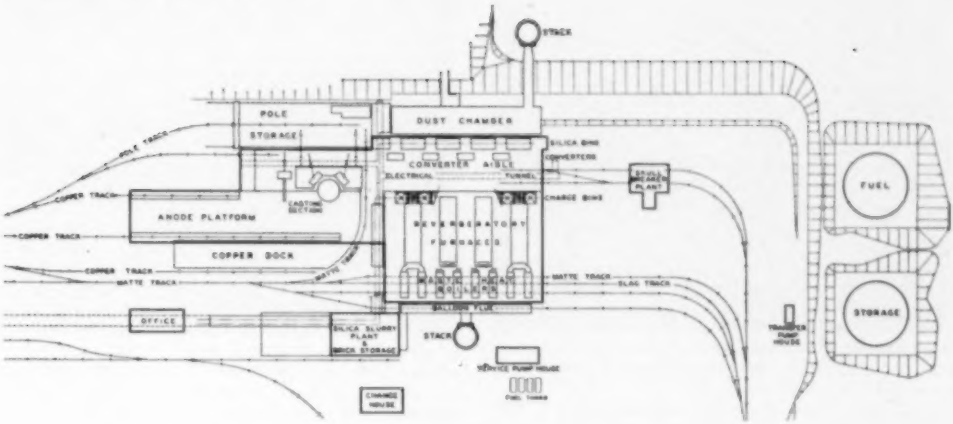
The bath depth is 3 ft. Matte is tapped through either of 2 tap holes in the flue-end wall of the furnace into short launders to fill matte ladles on special low-deck cars. Slag is skimmed through either side wall of the furnace, near the flue end, into longer launders to fill slag ladles in tilting cars. A ventilation system is installed over the tapping and skimming ports to remove the fumes. The matte and slag tracks are adjacent in a passage way at yard level below the waste-heat boilers.

Emergency tap holes are provided in each side wall of the furnace for use in the event of a break in the furnace wall. Trenches in the floor close by carry the molten metal under the matte and slag tracks to pool areas away from the smelter where it may be reclaimed when repairs have been made.

Each reverberatory furnace is fired by 12 oil burners mounted in groups of 6 on each side of the converter slag return launder. Bunker C oil is used. Blowers mounted on pedestals on the converter-aisle floor in a recess in front of the furnace foundations provide 25 pct of the required combustion air at 40 oz pressure. The remainder of the combustion air is drawn into the furnace through the burner ports and through pipe sleeves inserted in the upper part of the burner wall on each side of the converter slag return launder. Emergency burners, one to each side of a furnace, are provided along with separate facilities for furnishing combustion air required by them.

Exhaust gases leave each reverberatory furnace through a single 5x20½-ft outlet into an uptake which divides the gases for passage through the two waste-heat boilers provided for each furnace. The sloping floors and some parts of the side walls of the uptakes are self supporting but the roof and the balance of the side walls are of suspended construction. A portable, solid-brick damper carried on a trolley may be dropped into a slot in the uptake cross-over to cut out a waste heat boiler for repair.

Smelter Plan and Equipment:



SMELTER UNITS

BEDDING PLANT

Three beds 50x360 ft, 11,000 tons capacity each

REVERBERATORY FURNACES

Four 30x125 ft furnaces, 650 tons of charge per day per furnace
Balloon flue and 300-ft stack

WASTE HEAT RECOVERY

Eight waste heat boilers
Four turbo-generators of 7500 kw each operating at 725°F and 400 psig

CONVERTERS

Four Peirce-Smith 13x30 ft converters
Three hundred-ft stack

CASTING WHEEL

Two casting furnaces 13x25 ft, 175 tons capacity each
Casting wheel with 26 anode molds in either of two sizes or 52 blister cake molds

STRAIGHTLINE CASTING

One holding furnace, tilting type
Endless chain casting section

Additional hoppers are provided on a platform ahead of the furnace uptakes for the storage of silica which will be used for fettling in the furnaces beyond the normal charging zone if this is found necessary by the operators. Silica for this purpose will be delivered by battery powered trucks from the silica slurry plant.

Oxygen for lancing is available from a header with convenient outlets at each of the reverberatory furnaces. A central station, located in the silica slurry plant, provides facilities for putting 8 bottles of oxygen into service for lancing as well as storage space for additional bottles.

Clay for launder linings and silica for hot patching are ground and prepared in the silica slurry plant. The silica is pumped to agitator tanks located between each 2 reverberatory furnaces. A pipe loop around each furnace equipped with convenient outlets provides patching facilities to all parts of the furnace. Clay is delivered to the plant by battery powered lift trucks.

Slag ladles are of 225 cu ft capacity and the tilting ladle cars are drawn by battery and electric locomotives. Battery power is used within the smelter building and overhead trolley wires provide power beyond. The ladle tilting mechanism is controlled from the locomotive cab.

Converter Plant

The four Peirce Smith converters are each 13x30 ft, spaced on 64 ft centers which are staggered with the reverberatory furnace centers. The center of the converter is 17 ft above the converter aisle floor to permit the use of bulldozers for clean-up. The converter shells are of all-welded construction with the riding rings mounted on the ends. The converter lining is chrome magnesite (Kromag) brick throughout; the end lining is 27 in. thick and the shell lining is 18 in. thick at the bottom and 13 in. thick at the top. The thickened end linings permit the use of only 50 tuyeres although the shells were arranged for 52. There are twenty 2-in. tuyeres and thirty 1½-in. tuyeres. All have special ball valves with renewable seats and are of non-clogging construction. The swivel joint in the blast line is constructed to eliminate external thrust. The quick-acting blast gate is operated by an air cylinder for blast-charge control. The puncher's platform is counter-

balanced and is suspended by cables from above providing clear space below in either the high or low position.

The converter uptake hood is a special feature of the plant. It was newly designed to eliminate pockets where accretions might build up. The back wall slopes steeply so that the hood should be self cleaning. Copper shot or dust falling back within the hood returns into the mouth of the converter. A large counterbalanced gate opens high enough to permit pouring from the matte ladles or the pulling of collars when that becomes necessary. A jug-type damper and housing is installed on top of each converter uptake to insure the free opening or closing of the damper as well as for turning the converter gases into the brick dust chamber. The dampers are counterbalanced.

Silica for the converters is delivered to storage bins, each of 750 tons capacity, by a traveling tripper in the conveyor gallery above the bins which are behind the converters. The halves of 2 storage bins provide silica for each converter. Silica is fed to the converters through the side walls of the uptake hoods by means of small measuring bins which are in turn filled from the large storage bins. This permits the rapid charging of silica to either side of a converter overcoming the slowness formerly encountered when using Garr guns. Gates on the large storage as well as on the small measuring bins are operated by air cylinders controlled from the converter operating platform or cab. Facilities are provided so that Garr guns may be installed in the future if so desired by the operating staff. Provision has been made so that silica may also be loaded into boats for delivery by crane to the converter mouths.

Controls for the converters have been installed both on operating platforms between each converter and in individual cabs located on the reverberatory plant floor directly across the aisle from each converter. The cabs provide unbroken vision for the converter operations and are air cooled for the operator's comfort.

One skimming platform is provided for each converter and this is of heavy concrete construction, the exposed surfaces of which are armored with steel grids. The converter operating platforms are of steel construction since they are not in the area where the battering by equipment in transit is to be expected. Only access walkways have been provided in the gallery behind the converters for servicing bin gates and counterweight drives.

Facilities have been provided for turning the converters and casting furnaces into position for emptying in case of a prolonged power failure.

Reverberatory furnace matte in 325-cu ft ladles is trammed into the converter aisle for delivery to the converters by overhead cranes. The ladles are equipped with 2 pouring spouts to eliminate the need for turning them when suspended from a crane hook. Special bails are provided for the large ladles. The arrangement of the converter skimming platforms affords more clearance for these ladles.

The converter aisle is served by two overhead cranes each with a 100-ton main hook and two 25-ton auxiliary hooks. The crane cabs are air conditioned and of the full-vision type. The crane rails are placed high enough above the converter aisle floor to permit handling of the large matte ladles to all points required.

Repair facilities for the cranes have been provided on an elevated platform at the west end of the converter aisle. A 10-ton repair crane operates on separate rails above the main-crane runway so that parts of the large cranes may be removed for repair or replacement when necessary.

Skull Breaking

A separate building, near the smelter, is equipped with a 50-ton overhead crane and a hammer bridge for the removal of skulls from the ladles and the breaking of the skulls for return to the smelter. Ladles with skulls will be transported from the converter aisle on cars to the skull-breaking plant. The overhead crane will handle the dumping of the ladles to loosen and remove the skulls into a hopper with a grizzly bottom. The hammer bridge rides back and forth over the hopper so that an air hammer may crack the skulls to sizes small enough to pass through the grizzly openings. Gondola cars or flat cars with boats will receive the broken material for return to either the smelter crushing plant or the converters.

Casting Section

Casting of anodes and blister cakes will be done in a section west of the matte track, within the smelter, where two casting furnaces are placed to suit the mold spacing and unloading positions of the casting wheel and one holding furnace is placed above the straightline casting machine. The furnaces are of the tilting type, 13x25 ft, and may be completely rotated for emptying into the pouring spoons. They are of similar design to the converters and the riding rings, ring gear, pinion gear, and drives are interchangeable with those of the converters. The furnaces are lined throughout with chrome magnesite (Kromag) brick backed with insulating brick to reduce heat loss. The bottom lining is 18 in. thick and the top lining is 13 in. thick. Each furnace is provided with an air atomizing oil burner in a hinged mount. Combustion air for these burners is supplied by 2 sets of blowers conveniently located near the casting furnaces. Separate air-cooled operating cabs are provided for each furnace. A poling platform 38x120 ft, of heavy concrete construction with armored surfaces, is provided around the two anode casting furnaces at the casting wheel.

Matte from the converters is delivered to the furnaces, which hold about 175 tons each, in the large matte ladles by the converter-aisle cranes. Anodes or blister cakes are unloaded on the anode platform area south of the converter aisle. The two anode casting furnaces at the wheel are equipped with extended charging spouts and poling ports and the holding furnace is equipped with a 6x6-ft charging door; burner ports and gas outlets are provided in all of the casting furnaces. Small ladles (each of 125 cu ft capacity) placed on self-propelled cars with remote control, are used for the removal of slag from the casting furnaces. Tracks for these cars extend from the north side of the converter aisle to positions which permit the spotting of a ladle beneath each furnace. Operation of the cars is controlled from panels on the columns across the converter aisle from the casting furnaces.

A battery-operated crane truck on the poling platform of each casting furnace at the wheel transfers poles from special bin cars in the converter aisle to the poling platform or inserts the end of a pole into the poling port on the furnace. A frame, over each of these furnaces, with rigging operated

One of the four 30x125-ft reverberatory furnaces before bottom pouring, shows burner-endwall bracing.

Converters are spaced on 64-ft centers, and staggered with the reverberatory centers on the main aisle.

from an air hoist is employed to handle the poles after they are placed in the furnaces. Charging racks on the poling platform assure maintenance of the crane truck batteries. Space has been made available, north of the converter aisle for the storage of 6 months' supply of poles.

A movable platform, remotely controlled, has been installed on rails to furnish access from the poling platform to the pouring spouts of the casting furnaces and to the deck of the casting wheel. When not in use this platform is spotted over the casting wheel near the columns south of the casting furnaces. Push button stations on the poling platform and the movable platform control the motion in either direction.

The casting wheel is of new design and is arranged to handle 26 anode molds in either of two sizes or 52 blister-cake molds. It has a flat deck giving greater working area around the molds. Stationary rollers support the wheel near the rim and a roller bearing is placed under the center. The driving motors are placed in a pit below the wheel and the drive pinions engage an internal gear bolted to the bottom surface of the wheel. Acceleration and deceleration of the wheel has a special feature of control incorporating a pilot flywheel. Operation of the wheel is so smooth that the wheel may be moved with no delay for cooling. The drive is reversible permitting the wheel to turn in either direction for the long or short travel of anodes from either casting furnace to the unloading position. A system of sprays above and below the molds has been provided for cooling. Operation of the wheel is controlled from either of the two cabs provided for the casting furnaces.

Copper is poured into the molds on the wheel from pouring spoons, hydraulicly raised or lowered, under the pouring spouts of the casting furnaces. The pouring spoons are of steel plate and lined with brick and a mud mixture. A built-in skimming bar holds back charcoal when copper is being poured. The tilting and levelling of the pouring spoons is controlled from the operating cab at each furnace.

Hydraulic rams raise the knockout pins in the mold when the mold reaches the unloading position. The wheel is locked in the unloading position while the anode or blister cake is being lifted out of the mold and may only start its travel again after the ram has been completely lowered. Bosh tanks are placed opposite the two unloading positions for anodes and two other unloading positions are on either side of these for the unloading of blister cakes. Controls for the rams are placed conveniently for access by the operator.

An overhead crane operating in an arc over the four unloading positions of the casting wheel is equipped with an air-cylinder-operated trolley from which an air-cylinder-operated hoist is suspended to remove the anodes from the molds and place them in either bosh tank or to lift the blister cakes from the molds to place them on pallets. The racks in each bosh tank will hold 38 anodes and, when filled, a rack is lifted from the tank by a 15-ton overhead crane and placed on the anode platform. Battery powered fork-lift trucks handle the anodes through weighing, tilting, and placing in storage on the copper docks. An electrically operated tilting device turns five anodes at a time from the vertical to a horizontal position for bundling prior to shipment in gondola cars.

The straightline casting machine is of conventional design, being an endless chain equipped with molds for the continuous casting of blister cakes. Control of the straightline machine, the holding furnace, and the pouring spoon is operated from the cab. Battery powered fork-lift trucks handle the cakes after they are dropped from the molds.

Most of the finished product from the plant will be loaded on cars inside of the casting section where two copper tracks have been located. An additional 15-ton overhead crane operates over this area. Copper dock space has also been provided outside of the smelter building.

Handling of Flue Dust

Dust which is collected in the reverberatory balloon flue will be transferred by a drag chain conveyor to a bin, thence to a tank from which it is blown to the silica slurry building where it is pugged and returned to the furnaces with the charge on the conveyor.

Dust from the converters will be collected in the bin bottoms of the brick-dust chamber by drag chain conveyors into a cross drag chain conveyor to a bucket elevator serving a bin in the converter aisle. Boats are filled from this bin and are then transported by crane to return the flue dust as cold dope to the converters.

Chuquicamata Sulphide Plant:

Smelter Stacks

Clarence W. Dunham
Consulting Structural Engineer,
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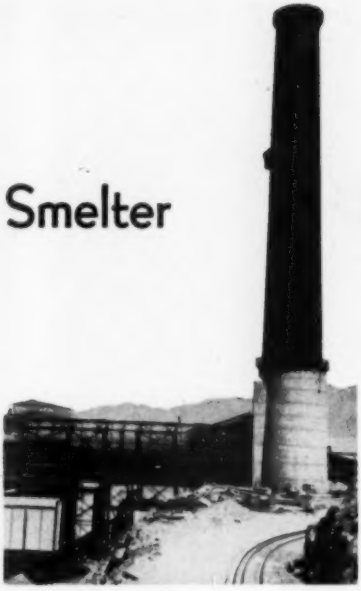


BECAUSE of the earthquakes that occur at Chuquicamata, the design of the smelter stacks constituted the most difficult structural problem of the entire sulphide ore project. Slight tremors occur almost daily and shocks of moderate magnitude are not uncommon. Naturally, the operation of the smelter depends upon the safety of these stacks, and their resistance to earthquakes was the critical feature of their design.

Adequate data of the magnitudes of past shocks at Chuquicamata were not available. However, considering the information that was obtained from experiences in other parts of Chile, a period of one to two seconds was adopted as a basis for the design.

To complicate the problem further, the stacks had to provide for operations under a wide variety of conditions ranging from the possibility of running one reverberatory furnace and two converters initially to perhaps five or six furnaces and eight or ten converters simultaneously in the completed plant. No one could tell for certain just what the ultimate capacity might have to be because the plans permitted a large extension beyond the first installation if this became necessary.

Studies of the stacks were made for many conditions. At first it was thought that one stack 600 or 650 ft high should be used initially for both the furnaces and converters, with a second similar stack to be built for the future extension. This height would provide adequate draft in the rarified atmosphere at the plant's elevation of 10,000 ft. It would also disperse the gases high in the air.

Preliminary studies of this large stack showed that such a structure was possible if made of steel but it would be exceedingly costly. Furthermore, the effects of the tendency for the top to whip under the action of earthquakes might be serious, and the design had to be based upon assumptions which might not be trustworthy. Such a tall stack was finally abandoned in favor of a larger number of smaller ones. Initially, one stack will service the first three or four furnaces. Another will handle the gases from four or five converters.

The smelter has been located in such a position that the prevailing winds carry the fumes away from the remainder of the plant. A height of 300 ft for the stacks was finally selected. The top diameter is approximately 24 ft 9 in. clear inside of the lining. The converter stack is located upon a hill alongside the smelter. This provides an effective increase in height because its base is approximately 66 ft above the yard level. The base of the reverberatory stack is at the latter elevation.

Stacks made of reinforced concrete or of steel were investigated, and so were many shapes for them. As these studies progressed, the lightness of a steel structure, and the ductility of the metal itself, led to the decision to use steel for the main part of the stacks.

The fact that serious damage to one of the stacks would cause the plant to shut down, caused special studies to be made of the stress conditions that might exist around the breeching which had to be nearly 15 ft wide and 40 ft or more high where it joined the stack. This height was necessary in order to provide adequate transition from the horizontal flow of the gases to the vertical direction without undue eddies. This opening constituted a serious source of weakness in the structure. If made with sufficiently strong reinforcement around its edges, there would be a hard spot in the structure that might cause trouble because of shaft vibrations.

Design Considerations

Some of the assumptions used as a basis for the design of the stacks are:

► Both stacks are to be identical if possible.

► Wind pressures are to equal 30 lb per sq ft on the entire projected area in side elevation.

► Lateral earthquake forces are to be based upon an acceleration of 0.1g for the bottom third, 0.15g for the middle third, and 0.2g for the top third of the height.

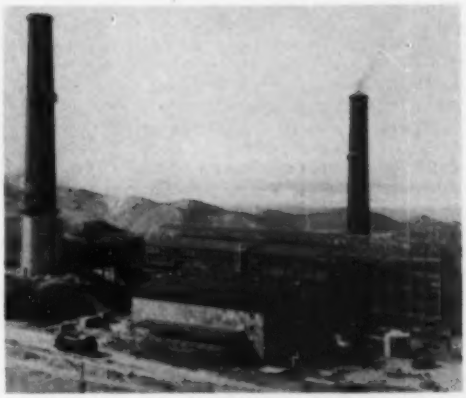
► The stacks are to be self-supporting, not guyed.

► Both stacks are to be designed for brick lining although it is not to be installed in the converter stack unless found to be necessary later on.

► Maximum soil pressure on the caliche or seamy rock is to be 6 tons per sq. ft.

► Safety factor against overturning due to wind or earthquake is to equal two. Both wind and earthquake need not be assumed to act at maximum values simultaneously.

► Design stresses in steel, concrete, and in reinforcement are to provide a safety factor of two.



Looking down at the smelter, power plant in foreground, converter stack at left, and reverberatory stack, at 66 ft lower elevation, at right.

The structure pictured in Fig. 1 was finally adopted as the basic design for both stacks. This is based somewhat upon the principles of the humpty-dumpty toy that has such a heavy weight in the bottom that it will right itself when tipped over. At least, it was found that a heavy, reinforced-concrete pedestal would be practicable and that the opening of 15x55 ft for the flue would not cause serious difficulty in such a massive structure.

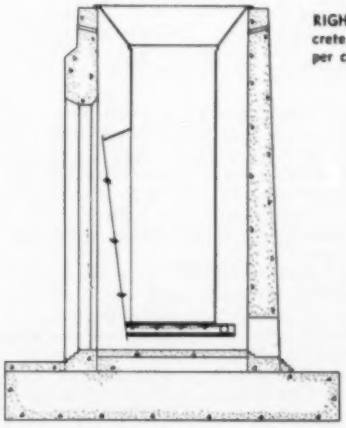
Pedestal Design

Some of the details of the concrete pedestal and the footing are shown in Fig. 2. A strongly reinforced edge and portal are provided around the flue opening. The heavy pedestal and the 70x10 ft octagonal mat cause the center of gravity of the entire 8000 ton structure to be relatively near the base. The safety factor against overturning is a bit over 5—considerably larger than the minimum required. This was done in order to have upward pressure under the entire base. The maximum soil pressure at one edge of the footing is approximately 7500 lb per sq ft, with zero load close to the opposite edge.

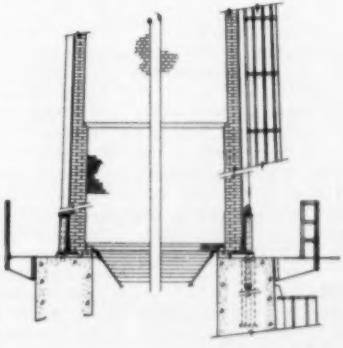
This concrete pedestal is separated from the flue. The latter, as pictured in Fig. 2, connects to a steel cone or hopper-like structure that rests upon bricks placed over the inner edge of the top of the pedestal. It is separated from the steel shaft above and is replaceable if this becomes necessary. It is insulated on the outside in the same manner as is the flue. The interior of the pedestal is ventilated by the holes shown near its top in Fig. 2 and by the openings near the bottom.

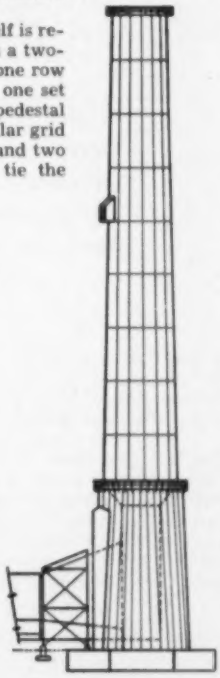
The 3-ft pad shown inside of the base of the pedestal and on top of the footing is a shear pad to serve as a key to resist horizontal seismic forces.

The octagonal footing itself is reinforced in the bottom with a two-way rectangular grid plus one row of radial bars. The top has one set of radial rods under the pedestal walls. It also has a rectangular grid across the circular center, and two rows of heavy dowels to tie the footing to the pedestal.



RIGHT: Fig. 1—General stack design. LEFT: Fig. 2—Concrete pedestal and footing. BELOW: Fig. 3—Detail of hopper connecting flue and pedestal.





The main shaft of the stack is steel with no openings of consequence in it. The shape is a frustum of a cone tapering from an inside shell diameter of 25 ft 7½ in. at the top to 35 ft at the bearing plates. This shape is unusual but it seems to be the best possible one to minimize whipping of the top. The plates were fabricated in approximately 25 ft lengths vertically, 15 of them forming a circle. Some details of the construction are shown in Fig. 3. The 6 in. WF-beam stiffeners, which are located at the vertical splices, give the steel shell considerable stiffness. The main plates are spliced horizontally by one or two plates instead of being lapped. Heavy angle-stiffening rings were used at the horizontal splices at the third points and at the top.

Shaft Design

In the analysis of the shaft, the shell and its stiffeners were assumed to be an undistorted ring. The vertical load of the lining was included for purposes of determining the extreme compression in the steel. The maximum compressive stress for the steel carrying the entire load is about 9000 psi., the tension, 3000 psi. Of course, the entire lining was also included when estimating seismic forces. The estimated period of resonance is about 0.8 sec.

Wind loads were investigated but were much smaller than those due to earthquakes.

The base of the shell is terminated in a girder effect that also serves as a boot for the anchor bolts. These bolts are replaceable. This was thought to be a desirable precaution if condensation of acid were to occur and possibly collect at the base, causing serious corrosion. The bolts are attached to embedded angles as pictured in Fig. 3.

The lining of acid-proof bricks is laid in Sauereisen cement mortar, and about ⅜ in. of this mortar is between the bricks and the steel shell. This allows for the thickness of inside splice plates where they are used. The rivets in such plates are countersunk on the inside. Elsewhere, the rivet heads are flattened to ⅜ in. The lining is 4½ in. for the top third, 9 in. for the middle third, and 13½ in. for the remainder. The top is protected by an annular ring of replaceable cast iron segments.

Estimates indicate that the thermal expansion of the lining and the steel shell will be somewhat similar. The bonding of the mortar to splice plates and rivet heads will tend to keep the two together. However, space is allowed at the top if the lining expands more than the shell.

The converter stack will have only the bottom portion of the lining installed at first. This part is needed to protect the concrete from heat and possible condensation of acid.

The final design was made by L. A. Warner under the supervision of A. B. Piver and the author.

Chuquicamata Sulphide Plant:

Piping

If the blood of Chuquicamata is copper, pipes are its veins. Thousands of feet, all sizes and shapes lace their way through the operation, carrying water, milk of lime, fuel, and compressed air.

J. P. Manning, Jr.
Design Engineer,
Anaconda Copper Mining Co.

UNQUESTIONABLY, the outstanding feature of the piping for the sulphide plant is the large amount which had to be done in almost every size from instrument tubing to 84 in. OD pipe.

In this article the piping will be considered under the main headings of Yard Piping and Building Piping. Tables I and II give data concerning the individual systems. In the case of yard piping the figures pertaining to lengths and weights are approximately correct; those for building piping are given only for the concentrator and smelter buildings where the amounts were the most extensive.

Yard Piping

The installation specifications for yard piping called for all sizes larger than 12 in. to have welded joints made with outside welding rings. This type of joint was selected over butt-welded joints because less time would be needed for lining up, damaged pipe ends would be less of a problem, and deflections up to 6° per joint could be made when necessary. These advantages were thought to outweigh the disadvantage of the greater amount of welding required.

Yard piping in sizes of 3 to 12 in. was specified to have butt-welded joints made with inside welding rings which were intended to facilitate lining up and keep welding material out of the lines. Screwed connections were specified for sizes under 3 in.

No protective coatings or wrappings were required on pipe to be buried because of the dry climate at Chuquicamata.

All piping crossing plant yards was specified to have a minimum cover of 30 in. to minimize danger of damage to lines and, for the equally important reason, to avoid restricting access in yard areas. As much as possible, all pipe runs were grouped in common trenches, with the exception of sewers.

The main plant is completely encircled by the Salado water and fire protection mains. Both large loops are about equally divided into two smaller ones by a north-south cross connection at the filter plant. Each large loop is over 10,000 ft long. Somewhat less extensive systems of piping are installed for distributing fresh water and compressed air.

Domestic wastes are collected by a system of concrete pipe sewers which empty into the tailing ditch well away from the plant to the south and are carried to final disposal in the Salar de Talabre.

The 10 million gal plant reservoir, fed by the Arroyo Salado pipe line is the source of water for the Salado water distribution, fire protection, and smelter power plant condenser water circulating systems. Makeup water to the concentrator head tanks is also supplied through a 24-in. pipe connected to the reservoir's 60-in. outlet manifold. Flow is by gravity to all these systems. That to the condensers in the smelter power plant is continually returned by pumping. The reservoir, when full, maintains 165-ft static head on the mains it feeds in the uppermost main plant yard. The head on the fire protection mains is further protected by connection to tanks having 400,000 gal of emergency storage capacity.

The minimum quantity of water available to systems supplied by the reservoir totals 35,000 metric tons per day, or an average of 6420 gpm.

Fresh water is in critically short supply at Chuquicamata. Consequently, its use is limited to drinking supply and the few requirements for which the somewhat brackish Salado water would be definitely unsuited. The storage tank for the fresh water distribution system is located north of the crushing plants at a sufficient elevation to supply all buildings by gravity flow. This tank, in turn, is gravity fed from others about 2½ miles away at the old oxide plant.

The system of piping which distributes compressed air to all principal buildings is supplied from a large receiver fed by two compressors in the smelter power plant. The rating for each of these units when operating at 187 rpm is 4785 cfm of free air. They are both driven by 800-hp electric motors allowing for maximum compressor discharge pressures of 125 psig. Auxiliary receivers are installed in the smelter and concentrator buildings.

Reclaimed Water System

The reclaimed water system is an important one as it is the source of roughly 75 pct of the water needed for operation of the plant. Gravity drain lines carry the clear water which overflows from all thickeners as well as miscellaneous drainage that is collected from the concentrator, smelter, and smelter power plant to a 1 million gal concrete tank that is sunk in the lowest plant yard between the tailing thickeners. Reclaimed water pump house No. 1 built on the slab covering this tank presently has six pumps installed with provisions made for two future pumps. The pumps are 400-hp electric motor driven, oil lubricated, vertical turbine units, designed to deliver 5000 gpm each when working

Table I. Sulphide Plant Yard Piping

System	From	To	Joints	Pipe		
				Size Range	Length in Miles	Weight in Tons
Salado water	Plant reservoir	Various plant buildings	Welded & screwed	1½"-12"	3.05	393.7
Fresh water	Oxide plant Toconce water tk.	Various plant buildings	Victaulic & welded	1½"-6"	5.39	239.8
Concentrator h'd. tank make-up	Plant reservoir	Concentrator head tanks	Welded	24"	0.81	170.9
Reclaimed water	Reclaimed water Pump House No. 1	Concentrator head tanks	Welded	36"	0.59	194.4
Concentrator mill water	Concentrator head tanks	Supply headers in concentrator	Welded	12"-36"	0.40	89.0
Condenser circul'g. water supply	Plant reservoir	Smelter power plant	Welded	42"	0.44	161.0
Condenser circul'g. water return	Smelter power plant	Plant reservoir	Welded	42"	0.36	133.0
Fire protection	Plant reservoir & emergency fire t'ks.	Hydrants, foam & sprinkler system	Welded	6"-12"	3.17	363.4
Foam fire protect'n. fuel oil stor'ge. t'k.	Foam system pump house	Oil storage tanks & foam hydrants	Welded	2½"-10"	0.64	26.2
Middlings thickeners overflow	Overflow from middlings thk'rs.	Tank at reclaimed water pump house	Welded	24"	0.15	31.6
Concentrate thickeners overflow	Overflow from conc. thk'rs, smelter & p.p.	Tank at reclaimed water pump house	Welded	8-24"	0.82	98.3
Head tanks overflow	Head tanks	Connection with middl. thk'rs. overfl.	Welded	18"	0.26	41.3
Compressed air	Receiver at power plant	Various plant bldgs.	Welded	1½"-8"	1.86	71.0
Fuel oil	Tank car unloading station	Smelter & lime burning plant	Welded	2½"-14"	0.84	43.0
Milk of lime transfer	Lime hydrating plant	Storage tanks north of concent'r.	Dresser	10"	0.55	58.2
Concentrate sand pumps discharge	Concentrator east end	Concentrate thickeners	Victaulic	5"	1.25	66.6
Miscellaneous	Various	Various	Welded	2"-10"	1.50	48.8
Sewers	Various plant bldgs.	Tailings disposal ditch	Cement mortar*	4"-15"	2.54	—
				Totals for yard piping	24.22	2121.0

* Concrete pipe, others steel.

Table II. Concentrator Building Piping

System	From	To	Joints	Pipe		
				Size Range	Length in Miles	Weight in Tons
Building service	Compressed air & water mains	Service connections	Screwed & welded	½"-4"	1.93	17.8
Flotation air	Air blowers	Flotation cells	Welded	8"-30"	0.25	25.6
Flotation reagent distribution	Reagent storage building	Reagent feeders	Scrd. weld. & victaulic	⅜"-6"	2.13	12.8
Lubricating oil	Sump tanks	Grinding mill bearings	Screwed	¼"-6"	1.92	25.3
Milk of lime	Storage tanks	Feed points & recirc'n. to reagent bldg.	Welded & victaulic	2"-6"	0.75	21.3
Mill water distribution	Head tanks	Grinding & flotation sections	Scrd., weld. & victaulic	½"-36"	1.83	79.1
Sand pump discharge	Sand pumps	Thickeners & flotation feed	Welded & victaulic	3"-8"	2.34	149.4
Thickener discharges	Thickener discharge spigots	Sand pumps & tailings ditch	Screwed & victaulic	1"-8"	1.65	124.4
Miscellaneous	——	——	Scrd., weld. & victaulic	½"-30"	0.68	12.2
Smelter Building Piping						
Building service	Compressed air & water mains	Service connections	Screwed	½"-6"	1.00	27.4
15# converter air	Smelter power pl't.	4 converters	Welded	24"-84"	0.11	62.8
Fuel oil	Service pump house	Reverbatories & converters	Welded	½"-8"	1.25	13.8
Reclaimed water drainage	Casting wheel	Reclaimed water system	Welded & dresser*	4"-14"	0.19	23.2
Miscellaneous	——	——	Screwed	¼"-6"	1.83	12.9
* Steel and C. I., others steel.		Totals for piping in concentrator & smelter buildings			17.86	610.0

against a 240 ft total dynamic head. All discharge through a 36-in. line leading to the head tanks north of the concentrator where the water re-enters the mill circuit. Pumps will normally be under the automatic control of switches which are set to vary the number operating in accordance with changes in the tank water level but provision is also made for manual control. One pump is always on standby.

The other important yard pipe line is the one for transferring 6 pct milk of lime. The 10-in. line starts from 5-in. sand pumps in the lime hydrating building at the southeast edge of the plant and runs diagonally across several plant yards to three steel storage tanks situated on a shelf above the concentrator reagent storage building, 2850 ft away. The requirements imposed for running this line were: 1—that a continuous slope be maintained toward the hydrating plant so that the line could be flushed at the end of each pumping period; 2—that it be made as accessible as possible for cleaning, repair, and periodic inspections; 3—that the number and degree of all bends be held to a minimum. To meet the requirements manholes were provided at every change of direction and at maximum intervals of 450 ft along straight runs. The line was continued unbroken through manholes but flanged sections at each were provided to allow for inspection and for the introduction of cleaning tools. Dresser-type couplings instead of welding for joints was used to give accessibility throughout. It was necessary to bury the pipe in a trench 30 ft deep at the point where it crosses under the main electrical tunnel running between the smelter and the concentrator.

Concentrator Building Piping

Service headers carrying compressed air, fresh water, and Salado water were run the entire length of the building along each main row of columns. Plugged outlets provided in each line at 50 ft intervals for the operators convenience supplemented connections to known points of usage.

A total of 55 sand pumps are installed in four pump pits ranged along the south wall of the building. They handle the transfer of all pulps and are of the overhead V-belt driven type with stationary adjustable sheaves. Pump discharge sizes vary from 2 to 5 in.; capacities from 80 to 875 gpm; percentage of solids in the pulp by weight from 18 to 50 pct. A standby unit and a spare is installed in each operating group. Maximum flexibility of operation is assured by the installation of a separate discharge line from every pump, making it possible to cut in the standby or spare as soon as trouble develops whether it be with the pump or its piping. This arrangement required 12,350 of schedule 80 pipe for sand pump discharge lines alone, but is considered excellent insurance against production losses due to pumping failures. Victaulic couplings were selected for the joints in long runs because these allow pipe wear to be equalized by periodic rotation and simplify line replacement. The only valves used were in the suction line to each pump and were of the rubber pinch type. The longest sand pump discharge lines are 1600 ft in length from the final concentrate pumps in pump pit No. 2 to the concentrate thickeners. Pipes were sized to maintain velocities within the range of 3 to 7 fps.

The reagents potassium xanthate, Aerofloat, and pine oil are piped for gravity flow from the reagent storage building to the concentrator. There they feed into three compartmented tanks mounted on as many columns along building line "C", each serving four of the concentrator units. The function of the tanks is to maintain the proper head at the reagent feeders supplied in the grinding, flotation, and regrinding sections.

Six pct milk of lime is pumped from a feed tank in the reagent storage building by a 4-in. sand pump

discharging to a 6-in. line that encircles the entire grinding and flotation sections of the concentrator. The rate of pumping maintained is about twice that of maximum usage with the excess being recirculated back into the feed tank for the purpose of always keeping a high enough velocity in the header to discourage the settlement of solids. Makeup to the feed tank is by gravity flow from storage and is controlled by a float valve at the feed tank.

Water for metallurgical operations in the mill is supplied from two headers running the full length of the building; one, between the rod and ball mill sections; the other, between the ball mill and flotation sections and both are fed by a 36-in. line from the concentrator head tanks with water that is a mixture of reclaimed water, containing some residual reagents and makeup from the reservoir. A third header, much smaller, is run along column line D to supply the regrinding sections with reservoir water from a separate head tank.

Smelter Building Piping

Service piping in the smelter is similar to what has already been described for the concentrator.

Air is supplied to the converters at 15 psig from the smelter power plant by four turbo-blowers, each with a free air rating of 30,000 cfm when operating at 1500 rpm. Air is brought to the converters by welded steel line which graduates in size from 84 in. OD at the smelter power plant to 24 in. OD in the branches to each of the four converters. Flow to the individual converters is recorded and indicated by watt-hour meters mounted to be read from the puncher's platform.

The remaining extensive system of piping in the smelter brings fuel oil to the 48 reverberatory furnace burners, plus occasional small amounts to holding burners in the converters and tilting furnaces at the casting section. Pumps in the service pump house, located south of the smelter building, take suction from service tanks in which steam coils keep the temperature of the oil near 120°F. The pumps discharge through heat exchangers in the same building which are supplied with saturated steam at 100 psig to raise the temperature of the oil going to the smelter to 200°F. To prevent any appreciable drop in temperature the oil lines are steam traced to each furnace and a portion of the flow is constantly recirculated back to the service tanks cooling by stagnation in the feeders. As a further precaution the oil supply, steam, oil return, and condensate lines are group insulated. When the smelter is operating at its full present capacity the fuel pumps will circulate Bunker C oil at the rate of 100 gpm.

Chuquicamata Sulphide Plant:

Electrical Distribution

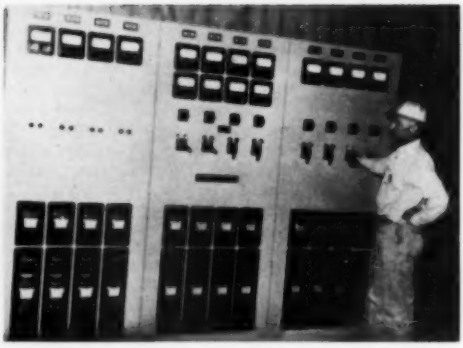
With existing electrical system providing the root for sulphide plant power, the new waste-heat generating plant was dovetailed to provide flexibility and emergency power for smelter in case of breakdown.

Stanley F. French
Chief Electrical Engineer,
Anaconda Copper Mining Co.

ELECTRICAL power and distribution for the sulphide ore plant is an addition and extension of the existing power system which has served the oxide reduction plant since its inception in 1914. The power plant is located at Tocopilla on the west coast of Chile, some 90 miles from Chuquicamata. A brief description of this plant and the existing transmission system is necessary to an understanding of the problems involved in the provision for the sulphide plant load.

Originally the Tocopilla power plant consisted of four Escher Wyss turbines driving Siemens Schuckert generators, each rated 10,000 kw. At various times, units of 10,000 and 20,000 kw were added, all of which operated at 215 psi and 600°F at throttle. In order to improve the economy of the station, a Topping unit was installed in 1938 together with a high-pressure steam generator of 360,000 lb per hr capacity. This unit is rated at 11,000 kw and was designed for 750 psi, 880°F. The exhaust from this turbine passes to the low pressure station header for operation of part of the low pressure turbines. Total kw generated by the high pressure boiler, topping turbine and associated low pressure turbines totals 33,000 kw and the total station capacity is 113,000 kw. This installation proved one of the most successful of its kind, operating almost continuously.

The original plant was of German design and consequently European characteristics were incorporated into the system, the frequency being 50 cycles and voltages of 500, 5000 and 100,000 were used for low, generator, and transmission voltages,



ABOVE: Main Chuquicamata substation receives power from Tocopilla. RIGHT: Modern enclosed-face switchboard typical of sulphide plant installation.

respectively. The generator voltage of 5000 v and is stepped up to 110,000 v at the high tension bus.

Power is transmitted to substation A at Chuquicamata over three transmission lines at the latter voltage. At substation A the largest portion of the load consists of rotary converters and synchronous motor generator sets which convert 50 cycle current into dc for use in the tank house. The use of so much synchronous equipment while permitting the operation of the system at a high power factor results in high system fault current, particularly on the 5000 v and 100,000 v busses at the receiving end.

Supply and Distribution Problems

In order to supply power for the sulphide plant it was necessary to extend the 100,000 v system to the new location and design an outdoor high tension station providing also for the power which would be later generated from the waste heat steam produced at the smelter. The main problems involved in the electrical power supply and distribution differed somewhat from normal plants of this nature. Apart from the aforementioned connection with the existing system, these problems were:

1—Design of the main system of distribution, which would restrict the short circuit kva at any point to breakers of economical size, but allow flexibility in operation.

2—Choice of voltage for the new requirements.

3—Provisions for flexible operation of waste heat steam turbogenerators, which under normal conditions of operation will generate approximately 40 to 50 pct of the sulphide plant requirements.

4—Provision for maintaining emergency power to the smelter area in case of system disruption.

A great many calculations were required to produce an economical solution to problem 1. This was due to the high potential rupturing capacity of the system. The short circuit kva on the high tension bus, due to the existing system, is over one million kva, to which the ultimate waste-heat plant would add an additional 750,000 kva if connected in a solid block. This value is also increased considerably by the synchronous load imposed by 40 rod and ball mill motors.

In designing power distribution in plants of this nature, it has always been our aim to divide the system along lines paralleling the material flow, the object being to minimize the percentage of plant shutdown caused by the failure of any single piece of main electrical apparatus such as high tension system breakers, stepdown transformers, bus sections, etc. While this system in itself naturally tends to reduce short circuit values at any point, further complication arose from the necessity of tying in the waste-heat generators to the low ten-

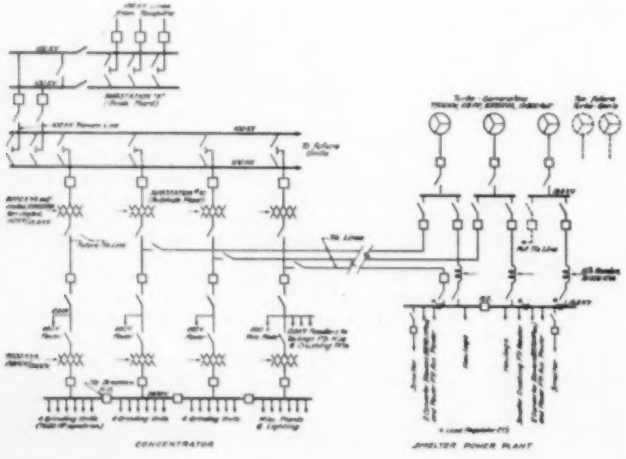
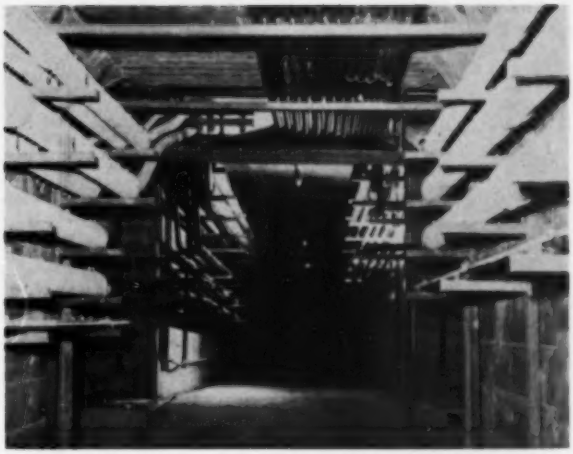


Fig. 1—It will be noted that generators are tied to the secondary of the main transformers through lines over which power flow must be reversible. Each main transformer supplies power through stepdown transformers to 2400-v bus section connected together by interlocked bus tie breakers.

Possibly the most interesting feature of power distribution is the tunnel system, which carries the major portion of the load. Main distribution centers are often located between or below main floors. Ease of installation, improved maintenance, and longer cable life were determining advantages of this system.



sion side of the main transformers. As in the main plant, design for the initial portion of the electrical distribution system must provide for ultimate conditions and for extension of the sulphide plant without disruption of the operating portion. While for the initial plant two 7500-kw turbogenerators were sufficient, provision for five ultimately had to be made.

The method of tying these units in to the individual transformers supplying the concentrator is shown on Fig. 1. From this it will be noted that the generators are tied to the secondary of the main transformers through tie lines over which power flow must be reversible. It will also be noted that each main transformer furnishes power through stepdown transformers to 2400 v bus sections connected together by interlocked bus tie breakers. These latter are for emergency purposes only and in normal operation remain open. By making individual concentrator and generator busses, the required rupturing capacity for the present and future conditions was held to 250,000 kva at the smelter power plant and 150,000 kva on the 2400-v system.

Selection of Voltage

The second condition mentioned above involved a decision which would either perpetuate the original voltages or establish others for the distribution in the sulphide plant. While it was considered feasible to continue with the secondary voltages of 5000 and 500 v, the adoption of voltages which have become standards on the American continent seemed advisable. For this system, 2400 v is best suited to distribution in the main concentrator and other plants where most motor drives range from 100 to 2000 hp. At the same time, it is not suitable for generator voltage except in the small size units, and for the greater distances 13,800 v possesses many advantages and is also standard. For this reason these two voltages were chosen. The latter voltage is also well suited to generators of this size and also provided an intermediate voltage to buffer the smelter power plant from the high short-circuit capacity of the high-tension system and considerably reduced the effect of the large synchronous motor load in the concentrator by interposition of transformers.

A. L. Boynton, superintendent of power plant, not only keeps the power flowing but also the hospitality at his base of operations, Tocopilla.

Roy Abbott is assistant superintendent, a Texan, and formerly employed by Anaconda in power plant operation at Cananea, Mexico.

The third condition involved provision for absorbing power into the system from the waste-heat turbogenerators, each rated 7500 kw, 0.8 pf, 13,800 v, at the same time making provision for supplying the loads in the power plant and smelter areas during the period when the concentrator is operated but no waste-heat steam is produced; or alternately in the case of loss of power from this source. This was accomplished largely by transmitting surplus power from the smelter power plant over the tie feeders with the main stepdown transformers operating at a normal rating of 8000 kva each. In event of turbogenerators being out of operation, the same transformers would then be able to furnish power to the smelter area by operating at 10,000 kva fan cooled. As additional units are added, extension of the main bus structure, duplicate sections, and additional turbogenerators may be installed without upsetting the system or interfering with operation.

Smelter Power Protection

Power to the smelter is provided in spite of possible system disruptions as mentioned in 4 above. Since outages are to be expected on the transmission lines, due to short circuiting by buzzards and from other causes, it was necessary to provide instant separation of the waste-heat generators from the existing system and from the concentrator load in case of a fault. If this were not done, the imposition of all of the sulphide and oxide plant loads would cause a sustained overload of 300 to 500 pct on the turbogenerators, resulting in total loss of power.

In the event of a major fault, therefore, provision is made to instantly trip the tie line breakers, thus leaving the turbogenerators free to supply emergency power to the smelter area. One difficulty which arises is due to the fact that in normal operation, the turbogenerators can only generate the amount of power resulting from the steam delivered by the waste-heat boilers. After the preliminary synchronization of generators, governors are responsive only to variations in the steam header pressure and power generated is proportional to the steam available from waste heat. At the instant of opening of the tie breakers these units without frequency or speed control would overspeed and be tripped out by the overspeed governor.

In order to maintain emergency power, however, a special form of governor has been developed, the position of which is controlled by totalizing wattmeter circuits in the emergency power busses. This device is known as a load regulator and at the instant of tripping of the tie breakers takes control of the turbogenerators then in operation and maintains their speed in accordance with the demands on the emergency load busses. Thus as long as steam is available power can be maintained to the smelter and restored to emergency pumping circuits within limitations imposed by the amount of steam available at the time.

In the event of total loss of power through accident, flywheel motor-generator sets which supply power to the converter tilting motors and smelter cranes are placed in the smelter substation proper. Each of these flywheels stores approximately 30,000 hp-seconds, which energy is available in emergency for tilting the converters to a safe position and the disposition of molten copper which may be in transit by the cranes at the time of failure.

Tunnel Distribution

Perhaps the most modern feature of power distribution in this plant is the tunnel system for the major portion of power distribution. In previous plants of this type, increasing concentration of power has made it necessary to improve the distribution system from the standpoint of ease of installation and maintenance, protection of important cable runs, and provision for adequate ventilation, especially in those locations where moisture can accumulate. As the main distribution centers and control rooms are often located between or below main floors, it has not been difficult to see the advantage in providing cable tunnels between the main outdoor substation and main control rooms.

This idea, while not new, was enlarged for the Morenci plant in 1939, where a longitudinal tunnel was added at the rear of the main concentrator control room, thus providing easy access to and installation of main cables to the switchgear located therein. The whole tunnel system, including the control room proper, was also ventilated by blowers and placed under slight pressure to keep out dust and prevent accumulation of moisture.

Secondary distribution, however, and cables beyond this point required duct and conduit systems under floors which are frequently wet and occasionally flooded. In the sulphide plant design, by locating the secondary low voltage unit substations and distribution centers on the lower side of the flotation section it became possible to extend the tunnels under the concentrator to this location and install the secondary distribution to the flotation regrind and pumping sections from the distribution centers, thus removing all conduits which would have been embedded in concrete to overhead and accessible locations. The obvious advantages during construction made this solution worthwhile.

A similar problem existed in the smelter, where all cables must be embedded in the converter aisle or run over high bays to various locations subject to heat and fumes, and the tunnel system was therefore extended to this section also. All of the main distribution, including 2300 v, 440v, and d-c feeders, is carried in the tunnel system from the two main control rooms at the smelter, the main tunnel being located under the converter aisle and protected from the possibility of damage from spills by a layer of brick and slag.

A tunnel also connects the main high-voltage substation with the smelter power plant and carries the tie feeders and main circuits between these two sections of the plant.

Ventilation of the tunnel system is obtained by suitably located blower units, which also discharge air through the tunnels into areas where clean ventilating air is required for switchgear enclosures and other equipment. The whole system is protected by closed circuit fire alarms which automatically shut down all blower units in the event of fire.

While the tunnel system is generally conceded to be considerably more expensive than a duct and conduit system, the factors in its favor, considering the expected life of a plant such as this, were overwhelming. Apart from the advantages in operation, the benefits gained in permitting the design of the buildings and structure to proceed rapidly, and the construction to be started long before details of power distribution could be worked out, have resulted in a considerable saving in time required for design and construction. In addition, due to the re-

sulting independence of the electrical construction crews, it has been possible to keep the electrical installation, which normally lags considerably behind building completion and equipment installation, well advanced in the schedule. As a consequence it was possible to energize circuits and provide power for testing as soon as equipment was installed and ready for test runs.

In conclusion, it can be stated that a considerable saving from the time required to get the plant into production has been realized. In addition, the use of this system has resulted in an installation possessing marked advantages from the standpoint of operation, expected length of life of cables, maintenance, replacements, alterations and extensions, when and if required.

Chuquicamata Sulphide Plant:

Waste Heat Power Plant

B. F. Koch

Steam Power Engineer, Anaconda Copper Mining Co.

COPPER reverberatories develop large amounts of exit gases of a temperature in the neighborhood of 2000°F. The gases are not only of a noxious nature but must usually be disposed of at considerable heights above the yard level by a stack. Handling these gases at the temperature mentioned would require rather expensive construction of conveying flues and stack. Moreover, wasting such amounts of heat is certainly most uneconomical.

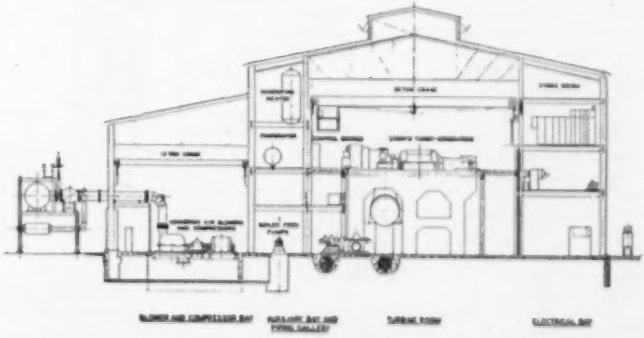
Numerous investigations have indicated that waste heat boilers of the water-tube type are the most practical and economical means for utilizing that heat although the gases are heavily laden with rock dust and copper-bearing slag, which upon cooling form hard deposits on the surfaces of dampers or heat absorbing elements placed in the gas stream.

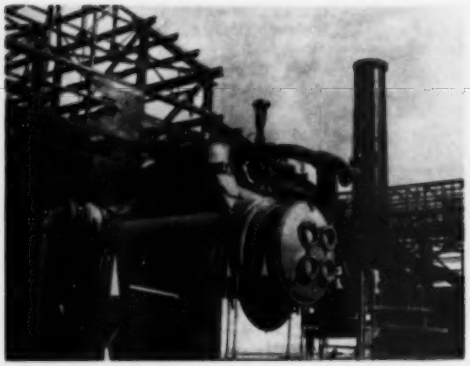
Avoiding Slag and Dust Deposits

Based largely on the experience of the paper industry in burning black liquor, a waste heat boiler design was developed a few years ago, providing not only extensive water-wall surfaces for cooling the gases below the fusing point of the slag but also settling out, as far as possible, the solidified particles of slag by creating low gas velocity. Matter carried in a stream of gases or air has a tendency to maintain its original direction upon a change in the direction of the flow of the conveying medium. Reverberatory waste heat boilers installed at Hurley and Morenci endeavored to utilize that tendency for the elimination of slagging particles at predetermined places and before they would strike the tube banks of the boilers.

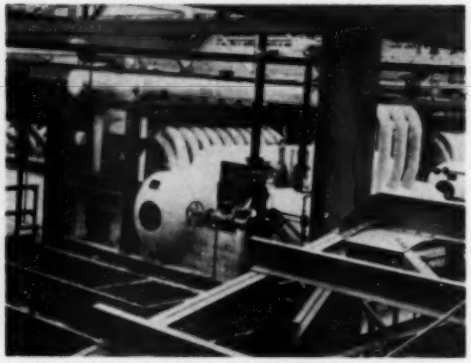
Both installations were successful in minimizing the slagging of boiler tubes and thereby minimizing the need for hand lancing. However, the low draft carried at the up-take of the reverberatory furnaces resulted in a positive pressure in the upper part of the furnace of the waste heat boilers, causing the emission of gases from lancing doors and making more difficult the already unwelcome work of cleaning the boilers by handlancing. Attempting to correct this nuisance as much as possible, the overall height of the Chuquicamata waste heat boilers has been reduced to the minimum considered necessary to obtain proper water circulation in furnace walls. Moreover, it is attempted to solidify slag and dust particles by providing large amounts of water-cooled surfaces before the gas stream reaches superheater and convection banks of the boilers. The furnaces have been made as deep as an economical and practical arrangement of headers for water

Waste heat steam is supplied at a temperature of 725°F to the throttles of the turbo-generators in the power plant. The waste heat power plant also houses motor-driven compressors for converter air supply.





Excess waste heat steam can be diverted to this outdoor steam condensing plant for water recovery. Condensate returns to feed water system ahead of deaerating heaters.



View within the waste heat boiler plant which produces steam at 400 psig and 750°F. Use of superheaters is minimized because of slagging tendency of gases handled.

walls and furnace roof would permit. The large water-cooled furnaces so created offer the added advantage of decreasing the velocity of the gas stream, thereby impairing its carrying capacity and forcing the heavier dust particles to drop out. Supported by the use of a number of wall-type steam-using slag blowers, the design has proved to be quite satisfactory in nearly eliminating the need for continual hand-lancing.

Before arriving at the decision to employ steam in the numerous slag and dust blowers installed on the waste heat boilers, the use of air-blowing equipment was investigated. Such investigation has proved to be inconclusive. Although the loss as condensate of the steam used is regrettable, it seems that the use of steam gives better results in breaking up the slag accretions. The steam lost in slag and dust blowers is being replaced by a bank of evaporators.

In line with Anaconda practice each reverberatory furnace is equipped with two waste heat boilers independent of each other, so that one may be cut out in case of tube failure without necessitating the shut-down of the reverberatory furnace. The waste heat from the reverberatory furnaces is sufficient to operate a steam power plant having the capacity to generate power for 40 to 50 pct of the requirements of the concentrator, mine and crushing plant.

Power Generation

The purpose of the waste heat power plant is not only to provide an outlet for the steam generated in waste heat boilers but also to form a source of emergency power in case of loss of supply from the Tocopilla power plant due to strike, sabotage or disaster. Furthermore, provisions have been made for auxiliary oil burners in the front walls of the boiler furnaces to permit their use as direct-fired boilers for furnishing steam to generate power entirely independent of Tocopilla supply and after a complete shutdown of the sulphide plant. These precautions appear to be obvious since the proper and uninterrupted functioning not only of plant lighting, fire protection, but also of hospital and town utilities is entirely dependent upon the supply of power either from Tocopilla or from the new smelter power plant.

On account of the additional cost of fuel due to freight charges from the coast to the mine, power generated in the waste heat power plant cannot compete with power generated in the company's Tocopilla generating station. Thus, separate direct-fired boilers are not justified and are not installed.

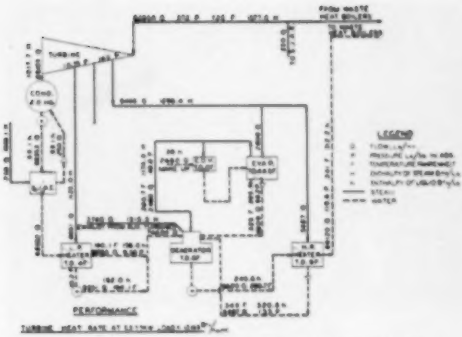
The waste heat power plant is of the conventional industrial type and the arrangement of its units may be considered typical of that used in power plants of the copper mining industry. Power generated at the waste heat power plant will replace that otherwise generated by steam direct-fired boilers at the Tocopilla power plant. Since all fuel consumed is fuel oil, imported and rather expensive, and since wages of operating personnel are not low anymore, more automatic controls and other refinements than usual are being employed in order to obtain good operating efficiency and to hold the labor cost to a minimum. In order to meet partial interruption of, or irregularities in, normal mine or concentration operations, flexibility of equipment has been striven for.

In view of the extreme slagging tendency of the gases handled, a superheating section is an undesirable feature in copper reverberatory waste heat boilers on account of the close spacing of its elements. Thus, superheaters are either entirely eliminated or their size is kept at a minimum. Except when used in combination with direct-fired boilers, as at Morenci, separately fired superheaters are uneconomical and could not be justified for the Chuquicamata plant. The temperature of the waste heat steam was therefore arbitrarily limited to a maximum of 750°F, resulting in a temperature of 725°F at the throttles of the turbogenerators.

Although higher steam pressures were investigated, a throttle steam pressure of 400 psig appeared to offer best returns on investment and was adopted. Furthermore, since the amount and temperature of the waste heat steam coming from copper reverberatory plants is dependent upon the charge and fuel ratio of the reverberatories and beyond the control of the power plant operators, it was considered advisable to provide automatic temperature control by installation of de-superheating equipment in the steam line to the power plant.

Turbogenerators

In parallel with the arrangement of concentrating and smelting equipment on the unit system, the turbogenerators were chosen of a capacity to each



Flowsheet gives pressure, temperature, and enthalpy data for waste heat power plant. If power consumption is curtailed steam goes to disposal condensers.

Converter air is supplied by these four motor-driven turboblowers of 30,000 cfm capacity each. Blowers are direct connected through speed-increasing gearing to induction motors, and operate at constant speed and pressure.

Four 7500 kw turbogenerators are installed. Capacity was chosen to absorb at best load amount of steam generated by one reverberatory furnace, in line with the arrangement of concentrating and smelting equipment on the unit system.

absorb at best load the amount of steam generated by one reverberatory furnace under expected normal operation, charge and fuel ratio. Turbogenerators of 7500 kw nominal capacity each are being installed in the waste heat power plant. However, a change to future units of 10,000 kw size can readily be accomplished without alterations.

Condensers

Each turbogenerator forms an independent generating unit with its condenser of 8500 sq ft, a two stage-twin steam jet ejector, a turbine-driven circulating water pump of 10,850 gpm, a turbine-driven and a motor-driven condensate pump of 160 gpm each, with air coolers and lubricating oil coolers being component parts of generator and turbine respectively. A steam hog common to all units serves for the quick evacuation of air from condensers. Each condenser is of two-pass design with a welded steel shell and cast iron water boxes. Its 8500 sq ft surface is made up of 16 BWG 1-in. OD, 18 ft 3½ in. long American Brass Company's Ambraloy 927 (Aluminum Brass) tubes, rolled in at both ends. Tube expansion is allowed by diaphragm expansion joint between one tube sheet and shell. Condenser cooling water is supplied from the mill supply reservoir under static head and returned to the reservoir by the circulating water pumps. Natural cooling in the reservoir is being relied upon for the initial plant but a cooling tower adjacent to the reservoir has been provided to care for the requirements of the ultimate plant.

Steam bled from two points of the turbogenerators is used in low and high pressure heaters for heating the boiler feed water to a final temperature of 360°F. Intermediate heating and deaerating is accomplished by vertical deaerating heaters with steam supplied by turbine-driven auxiliaries. Eight-stage turbine-driven centrifugal pumps, one of which is equipped with dual drive, pump feed water to the boilers under three-element automatic control.

Evaporators are installed for providing make-up water for normal operation, but connections have been made for the emergency use of fresh water. Raw water for evaporator feed has a high salt content which necessitated the installation of chemical feed equipment and heavy evaporator blowdown to control steam purity. Distilled water is stored in an outdoor storage tank and the flow to and from the system is regulated through a manual control station.

While the generation of waste heat steam will be continuous, the consumption of power on scheduled weekend shutdowns or during a turbogenerator outage will curtail the use of waste heat steam. During these periods excess waste heat steam can be diverted to an outdoor steam condensing plant for recovering the water. This plant consists of a surface condenser of a size duplicating those serving the turbogenerators, of circulating water and condensate pumps, also control equipment for automatic regulation of steam to be condensed. The condensate from this waste heat steam disposal plant is returned to the feed water system ahead of the deaerating feed water heaters.

Plant air at 100 psi is produced in two 4785 cfm electrically-driven compressors. For the copper converters four motor-driven turboblowers of 30,000 cfm capacity each are provided. The blowers are direct-connected through speed increasing gearing to induction motors and operated at constant speed and pressure and are equipped with necessary automatic controls for this type of operation. The air compressors and converter air blowers are located on the ground floor of the waste heat power plant in the space normally occupied by steam generating equipment of a conventional power plant. Considerable saving in operating and maintenance expense has proved the value of such arrangements at other smelters.

Chuquicamata Sulphide Plant:

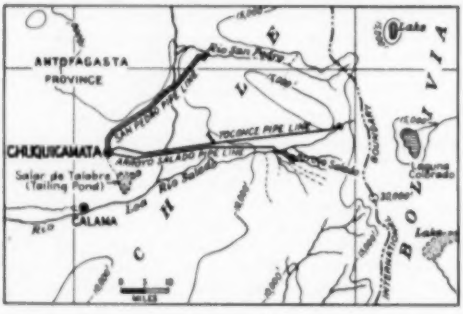
Water Supply

W. E. Rudolph and R. E. Baylor

Chief Engineer and Asst. Chief Engineer,

Chile Exploration Co.

DUE to its location in the Atacama Desert, one of the most barren of the earth's surfaces, Chuquicamata's water supply presents unusual problems. Yearly rain-fall averages less than one tenth of an inch at the plant. However, there are summer showers above 12,000 ft in the Cordillera to the east, the resulting run-off flowing through old river valleys buried beneath more recent volcanic formations, to be impounded within sediment-filled basins. This water emerges at springs where the outlets of these basins are blocked by lava flows, and here are formed the small streams which feed the only important river of the region, the Rio Loa. Chuquicamata's water is obtained from these springs and rivulets.



The map above indicates four pipe lines from which potable and industrial water are supplied. Potable water, amounting to 4500 metric tons per day, is conveyed in the Toconce pipe line from springs 59 miles due east of Chuquicamata. This water is used not only for drinking, but also for boilers and other needs requiring high quality. For industrial water at the oxide plant, there are two 12-in. pipe lines from the Rio San Pedro, carrying a total of 17,000 metric tons per day of slightly brackish water. This water is at present used mainly for leaching and for hygienic purposes.

Water Source Found

For the present and future needs of the sulphide plant, it was calculated that at least 32,000 metric tons per day of make-up water would be required. For this purpose, a pipe line of 44 miles length was constructed to bring in the entire flow of the Arroyo Salado, one of the eastern tributaries of the Loa. The salt content of this water is so high (over 5000 parts per million of solubles, mostly chlorides) that it is highly detrimental to farming, and the Chilean Government had been studying projects to separate these waters from others of the Loa system in order to improve agricultural conditions in the fertile valley of Calama. So it happened that the Government was willing to award rights to the Arroyo Salado waters under agreement whereby the Mining Company removes waters from the Rio Loa system above Calama for all time. The outlet of these waters, after serving their purpose at the new concentrator and leaving the plant in tailing, is the Salar de Talabre, an old salt lake which presents fully ten square miles of surface to serve as an evaporating pan, the outlets having now been blocked by dams. Here the dry climate of Chuquicamata is a favourable factor, evaporation averaging slightly above ¼ in. per day.

The Toconce and San Pedro pipe lines have been functioning from 26 to 34 years, and through the use of special cleaning tools which were developed at the plant, as well as deaeration of the more active Toconce water, these pipes are now maintained at capacities which do not diminish as years go on.

Constructing the Dam

The Arroyo Salado pipe line design and construction involved certain special and interesting features, and inasmuch as this line and its intake works are solely for the needs of the new sulphide plant, more detailed description is given. The waters are impounded at a gravity dam constructed of concrete to a height of 100 ft above the river bed, keyed into the precipitous Dacite walls of the narrow canyon (barely 6-ft wide at the bottom, only 25-ft width at 50 ft above). A small secondary dam was built 100 ft down stream from the main dam, providing a pool of 15-ft depth to protect the main structure from flood flows over the spill-way during the rainy season. A system of four 36-in. syphons was designed for discharging these flood waters from the lower depths of the lake, in order to avoid eventual sedimentation behind the dam. The lake has a length of 3300 ft, and its water level is controlled by an adjustable spillway permitting draw-down of eighty inches, amounting to 41,000 metric tons of available capacity. This regulation is necessary because of wide fluctuations in stream flow between day and night due to freezing of feeders.

During the construction of the dam the entire river flow was handled within a 36-in. pipe line some 2000 ft in length. As the excavations proceeded

Syphons on dam (left) take water from bottom of lake during flood season thus preventing silting up. Over and up, describes the above-shown difficult section of pipe line. At the top it enters a 3600-ft rock tunnel.

to bedrock, 26 ft below river bottom, this pipe was hung by cables fastened to the steep walls of the canyon, a river suspended from above while the men were working under it. The 36-in. pipe within limits of dam construction was embedded within the concrete, and fitted with a valve at the upstream face to provide emergency means of emptying the lake if ever required.

Settling and Deaeration

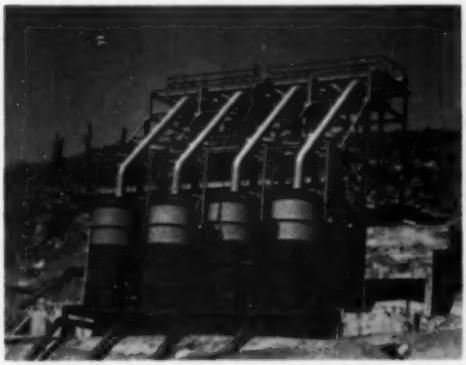
Leaving the lake through a 36-in. pipe with invert 18 ft below spillway level, the waters flow to four settling basins, each measuring 40x150x16 ft maximum depth, arranged in parallel. These will function only during the flood flows of the rainy season, inasmuch as the water of the Arroyo Salado is ordinarily clear, and what little turbidity does exist drops in the lake above the dam.

From the outlet of the settling basins the waters are conducted in a 36-in. pipe to a deaerating plant, where oxygen is removed in order to protect the pipe line from oxidation and deterioration. This installation consists of four deaeration units set in parallel, with first and second stage of each unit in series. For the first stage there is a 10-in. ejector, and for the second a 24-in. ejector, requiring 406 and 1800 gpm of water respectively for producing the necessary vacuum in the deaeration pumps. A total potential height of 147 ft, less friction head loss in the pipe, is available for operating the deaeration pumps, a total of about 2200 gpm or 12,000 metric tons per day passing through the vacuum ejectors of each deaeration unit. Thus the number of deaeration units in service may be adjusted to varying flows in the river.

The vacuum tanks of the deaeration plant, one for each pump, measure 10 ft diameter by 21 ft height, and are packed with wood strips. As evidence of the resourcefulness of engineers working in faraway places, these tanks (weighing over 13 tons each), as well as all the steel work of the deaeration plant (to 100 ft above ground surface), were erected with two tractors. The internal operating pressure at the first stage deaerator tanks is 1.5 in. of mercury absolute, and that at the second stage tanks 0.35 in. abs. Whereas the water entering the deaerators contains 9 ppm of dissolved oxygen, that leaving the plant contains none at all. Lacking this oxygen to attack the steel, it is expected that his pipe line should function indefinitely without appreciable diminution in capacity.

The Pipe Organ

The line leaving the deaeration plant is 30-in. diam., with later sections of 26, 24 and 22-in. diameter to fit hydraulic grade and internal pressures to conditions exacted by the profile. The neces-



The first of its kind erected, the deaeration plant will protect pipe line for years to come from deterioration. It worked properly from the day it started.

A third hurdle, the settling basins which function in the rainy season, only, make it practically impossible for silt to get into pipe line.

sity of carrying a wide range of river flows without admitting air developed a device for choking capacity, known as the Pipe Organ. This consists of a 6-in. auxiliary pipe laid beside the 22-in. main line for a distance of 800 ft, with eleven 6-in. ladder pipes, each with its shut-off valve. By closing the valve on the 22-in. pipe, and regulating the flow through a greater or lesser length of 6-in. pipe, flows ranging from 49,000 down to 24,000 metric tons per day can be handled at C-140 in the Hazen-Williams formula, and in similar proportion for smaller values of C.

The pipe line and intake works were constructed by the company's own forces, aided by Chilean contractors who furnished labor on a piece work system. The pipes were delivered in 40-ft lengths in nests of 4 or 5 pipes of the different sizes, with specially designed wood caps for holding and protecting the ends of the pipes within the nest. The line was laid above ground for the great part, slip-on couplings being used excepting adjacent to bends, where connections were welded. The pipe was covered between joints because of wide variation between day and night temperatures.

During the early operations of building an access road within the Salado canyon, the pioneer surveyors and construction men had nothing but grass roots for support upon rugged rock slopes. These precipitous canyon walls made any pipe line vulnerable to rock slides, and to avert extremely bad conditions at one location the line was passed through a tunnel 3600 ft in length for safety.

Chuquicamata's long distance pipe lines are maintained by a crew of 22 men, and whereas reservoirs and storage tanks provide a day's capacity for all purposes, most breaks are repaired and water service resumed within eight hours after alarms on pressure gauges indicate that something is wrong. A separate group of 6 to 8 men service the distribution systems at the plant and housing areas.

The Oxide Plant

E. V. Herring and J. C. Allen

General Superintendent and Asst. General Superintendent, Oxide Plant, Chile Exploration Co.

CHILE EXPLORATION CO's. oxide plant at Chuquicamata, the largest of its kind, started in 1915 with initial operations at the rate of 10,000 tons of ore per day. Anaconda Copper Mining Co. acquired the controlling interest in 1923 and in the next few years modernized the plant and mine, extending operations to a nominal capacity of 490 million lb of electrolytic copper yearly by 1942. Plant production averaging 425.7 million lb of copper per year was maintained for the eight-year period extending from 1941 to 1948 inclusive. During the World War II production reached its highest level when in 1944 the output was 531.8 million lb. The maximum production for any one month was 49.6 million lb, achieved in May 1947.

E. A. Cappelen Smith, consulting metallurgist for M. Guggenheim's Sons, worked out the first process for the treatment of Chuquicamata copper oxide ore about 1913, and directed a staff of 20 young engineers operating a pilot plant at Perth Amboy, N. J., on three shifts for an entire year.

The copper is extracted from the oxide ore by a hydro-metallurgical treatment described in detail by T. A. Campbell in AIME *Transactions*, Vol. 106, Copper Metallurgy, 1933.

The Oxide Plant is divided into five main operations as follows: (1) Crushing, (2) Leaching, including dechloridizing and sulphur dioxide treatment, (3) Tailing disposal, (4) Electrolytic precipitation, (5) Smelting and melting.

Crushing

The present Primary Crushing Plant has been in operation 25 years, during which time 4,342,112 cars of 70 tons net capacity have passed through the dumpers. The ore as received from the mine is frequently blocky and up to 5 feet in section. It is reduced to 9-in. size by two 60-in. gyratory crushers, and transported to the east and west ore bins by 60-in. conveyors.

In the secondary crushing operation the ore is first reduced to 2 to 3-in. size by five standard 7-ft cone crushers and two No. 10 gyratories. This product is then reduced to $\frac{3}{8}$-in. size by fifty-two 48-in. vertical Symons disc crushers and is transported by belt conveyors to the loading bridges from which it is charged into the leaching vats.

Conveyor belt No. 14 which serves the eight vats in the east section of the leaching plant has transported over 125 million tons of ore, which is a record for any type of conveyor. This 60-in. belt travels at a speed of 600 fpm and carries loads up to 5500 tons per hr.

Leaching

The ore is leached in 14 vats, averaging 13,500 short tons net capacity each. These vats are lined with a natural mastic which adequately resists the sulphuric acid leaching solutions.

The leaching process as presently conducted comprises soaking the ore with successive volumes of leaching solution and spent electrolyte and withdrawing successive volumes of enriched solution from the vat to a total of approximately 3.2 million gal. The first 1.6 million gal of this volume withdrawn is called strong solution and contains about 35 parts of copper per 1000. The balance of 1.6 million gal subsequently withdrawn is stored in a

solution sump and later withdrawn therefrom to cover a succeeding vat of fresh ore and to produce strong solution from it. When this intermediate solution is withdrawn from the ore, fresh spent electrolyte and solution advanced from the washes replace it. The ore following these soaking or leaching periods is then washed with a series of five wash solutions each containing decreasing concentrations of copper and sulphuric acid and is finally washed with a volume of 396,000 gal of fresh water.

Approximately 264,000 gal of moisture is absorbed by the unusually dry ore which contains about 1½ pct moisture as mined, and about 26,400 gal is lost in evaporation. The difference of 106,000 gal of fresh water is therefore advanced through the washes and into the primary leaching solution for each vat of ore treated, and a corresponding volume of 106,000 gal per vat is stripped of its copper and discarded to waste thereby effecting control of impurities, particularly iron, molybdenum, arsenic, etc. To further increase the efficiency of the leaching process an additional 106,000 gal of wash water per vat is passed through special electrolytic cells wherein a portion of its copper is removed and the solution returned to the washing system.

The method of leaching employed is a batch system and the total cycle time from the start of loading the ore into the vat to the termination of the excavation of the leached tailings averages about 72 hr. The Leaching Plant, therefore, has a capacity of approximately 4.7 vats per day.

The mineral antlerite, a basic copper sulphate, supplies the bulk of the sulphuric acid gained from the ore. This acid gain is currently 3.3 to 4.4 lb of sulphuric acid per metric ton of ore. To augment the amount of acid required to operate the leaching system sulphur dioxide is added to the strong solutions just prior to electrolysis, in amounts ranging from the equivalent of 0.44 to 3.1 lb of sulphuric acid per ton of ore leached. As the electrolyte contains nitric acid, chlorides, ferrous iron and molybdenum compounds leached from the ore, the sulphur dioxide added to the solution before electrolysis serves additionally to stabilize it by preventing decomposition of the nitric acid and self-oxidation of the electrolyte.

Recovery in leaching amounts to about 95 pct of the oxide content of the ore and 10 pct of the sulphide content. The average overall recovery on the 360 million tons of ore treated in the plant from the start to December 31, 1951, which contained 1.70 pct total copper, has been 83.75 pct. Though the nominal rating of the plant is 1.4 million short tons of ore per month, a total of 1,666,748 tons were treated in June 1952.

Chlorides, which are present in the ore and also in the water used for washing, are precipitated as cuprous chloride by agitating the strong solution with finely divided cement copper. The cuprous chloride is subsequently settled out of the solution and recovered. This process takes place before the solution is delivered to the electrolytic cells. The cement copper is regenerated by dissolving the settled out cuprous chloride in a ferrous chloride brine solution which is then passed over scrap iron in 7-ft diam bronze drums.

Tailing Disposal

The 14 leaching vats are served by five unloading bridges, employing clamshell buckets, which vary in capacity from 6 to 12 tons. Two bridges operate together on a single vat, and discharge at the rate of 12,500 short tons in eight hr. Approximately 800 tons of tailing are left to protect the bottom. The buckets discharge into hoppers built into the pier legs from which the tailing runs by gravity into 18-cu yd hand-operated dump cars or air-dump cars of 30 cu yd capacity. Four trains of 24 cars powered by 70-ton electric locomotives discharge the tailing over the side of the dump which is 4 miles in circumference and ranges from 8 to 330 ft in height.

Power is taken from third rail in the yards and on the permanent portion of the dump. The active track on the dump is equipped with a flexible trolley system which is carried on poles clamped to the under side of the running rails and follows the changing contour of the dump as the track is shifted. A new auxiliary dump built about halfway out and on top of the main dump allows for 10 million tons additional tailing storage.

Electrolytic Precipitation

The tank house building, 1187 ft long by 160 ft wide, is divided into three bays. There are 1098 mastic lined tanks in the building, each approximately 19x4x5 ft deep.

In the commercial cathode sections, 67,556 lead-antimony-silver anodes equipped with long mastic strip insulators are in service, and 5457 anodes of a similar type are employed in the commercial starting sheet sections. In general, 16 or 17 tanks comprise a section and eight sections in series electrically make a circuit, of which there are nine in the house. Each circuit is powered from one or more rotary converters or motor generator sets. Maximum loads up to 62,000 kw corresponding to 800 tons of deposition daily have been carried. Load carried per circuit averages 20,000 to 25,000 amp and 255 to 270 v; but 34,000 amp and 290 v can be carried on circuit 8. Current density averages 14 amp per sq ft. Anodes are placed on 3-in. centers. Magnetite, ferrosilicon and copper silicide (Chilex) anodes have all been used in the past, the latter having been replaced entirely by the present lead-antimony-silver anode.

Efficiencies average 85 to 92 pct with a yield of 25 to 28 lb of copper deposited per kw-day under normal conditions.

Commercial starting sheets deposited from strong solution with insoluble lead-antimony-silver anodes are used entirely.

The inflow solution to commercial cells is made up of starting sheet return and first stage spent electrolyte, part of which has been passed over the towers of the sulphur dioxide plant; and second stage-spent electrolyte averaging 15 parts of copper per 1000 is returned to the leaching plant.

Flow rates of solutions vary from 200 to 265 gpm. Temperature of the electrolyte is held between 36° and 30°C.

Neophrene sleeves are used on all anode tail bars and the development and use of the long mastic strip insulators has eliminated at least 50 pct of the inspection required in former years. Cathodes deposited to 150 lb are transported over the industrial railroad to the Furnace Department, where market shapes are cast.

Fourteen different electrolytic copper shapes are commonly cast by the furnace department including 10 sizes of wire bar varying from 135 to 310 lb,

end poured wedge cakes, flat cakes, billets and ingotbars.

Furnace Department

Three reverberatory-type market furnaces of 400 tons daily capacity having a hearth length of 49 ft 5 in., and width of 13 ft 10 in. are in service. The bottom and roof are of silica brick, and the side walls are magnesite lined. They are charged through three charging doors by Morgan open hearth charging machines.

The charge consisting of hard dense pure cathodes plus rejected shapes, warmers or skulls, is expertly packed into the furnace by two charging cranes, after which the furnace is sealed and the melting operation started. Fuel oil burned by two burners is gradually brought up to 460 gal per hr, and held at that rate until the end of the melting down period.

Boiling caused by the melting of cold metallics on the bottom of the furnace, takes place within 8 to 10 hr, after which the fuel oil is decreased and the charge skimmed clean. Air at 12 lb pressure is blown into the molten metal by iron pipes inserted through small openings in the door. This is continued until the granular texture of the test buttons indicates that approximately 0.8 pct oxygen is contained in the metal and that scorification is complete. The oxidized-silicate slag formed largely by erosion of roof brick and bottom is then skimmed off. After skimming the charge is covered with charcoal and the reduction of copper oxide by means of poles forced into the metal bath is started. When the oxygen has been reduced to .03 to .04 pct the furnace is tapped and casting starts. Each furnace is equipped with a 40-ft diam Clark-type mechanical casting wheel. All molds are made of refined copper and are replaced after four to seven charges. The cast bars or shapes are given a rigid inspection by two inspection crews and rejects are remelted. After the second inspection, the bars are weighed on the production bullion scales and prior to shipment are lot marked and re-weighed on the shipment scales.

Antofagasta Copper Yards and Shipping

A. J. Chellew

Robert Condit

Port Agent and Asst. Port Agent, Chile Exploration Co.

THE Company maintains a port agency at Antofagasta for dispatching copper and also for receiving from ocean vessels the vast variety of materials and supplies required at the plant. Copper shipments from the oxide plant refinery to Antofagasta amounted to 162,000 metric tons during 1951, while incoming freight totalled 175,000 metric tons.

Electrolytic market copper in various shapes is currently loaded at the plant upon 20 or 30-ton cars of the meter-gage Antofagasta-Bolivia Railroad, to be transported a distance of 165 miles to the company's yard at Antofagasta. Here it is unloaded by cranes at the rate of about 175 tons per hr. Copper not scheduled for immediate shipment is stocked in tiers within the yard, but most of it is reloaded and sent to port works or a wharf serving lighters.

Some of the vessels entering Antofagasta tie up at the port works, while others anchor inside the breakwater. In the latter case, all of the loading into the hatches is from 50-ton lighters towed to the vessel. When the vessel is at the dock, about half of the copper is loaded by port works cranes and the other half is loaded by ship's tackle from lighters.

Fuel Oil Requirement Expands

One of the innovations caused by the construction of the sulphide plant has been the necessity to provide facilities at Antofagasta for receiving and storing greater quantities of fuel oil than in recent years. From here it is loaded into tank cars for transportation to Chuquicamata. The older installation consisted of a 10-in. submarine pipe line, heating, and pumping facilities, a 67,500-bbl storage tank, and railroad loading rack, located at Antofagasta's northern limits. Two new tanks, of 80,000 and 30,000-bbl capacity respectively, have now been added and a second submarine pipe line will shortly be installed. Fuel oil requirements for the sulphide plant are estimated at 39,000 bbl per month, in addition to which 9000 bbl per month are required for the operation of the oxide plant. The company has a special lighter which is used by its diver for connecting the oil line with the oil tanker when disembarking oil, this lighter also being used for inspecting and overhauling buoys, as well as for recovery of bars of copper which are sometimes dropped into the bay when loading vessels during periods of rough weather. Transport of oil to Chuquicamata is by 76 tank cars of 20-ton capacity and 20 of 30-ton capacity, all assigned to the company's fuel oil shipments excepting six for Diesel oil and one for gasoline.

Incoming Freight

Other freight received by the port agency has been averaging about 4800 metric tons per month of operating supplies for the oxide plant (renewals of equipment, spare parts, lubricating oils, and greases, etc.), some 1900 metric tons per month for the company stores, and about 6000 metric tons per month of materials and equipment for the construction of the sulphide plant. The greater part of these shipments (particularly the operating and construction supplies) originate from abroad, and must be handled through the Chilean Customs. Hence the company maintains a Customs storage warehouse (goods pending appraisal of Customs duties) in addition to its other storage facilities at Antofagasta.

Not the least of the Port Agency's duties is to meet air and steamer passengers *en route* to Chuquicamata and to arrange for their trips up the hill. Landing in a strange country with a strange language, the first impressions of a new employee might otherwise be indeed confusing when facing Antofagasta's unique setting of brown barren hills.

THE DRIFT OF THINGS

(Never having been in Chuquicamata, this department is being turned over to the Editor of Mining Engineering this month. E. H. R.)

WHEN the new sulphide plant started to roll at Chuquicamata last July, we broached the matter of looking in on this historic event to Burr Wheeler, then executive vice president of Chile Exploration Co. (Chilex executive offices are housed with the parent company, Anaconda, at 25 Broadway which is 23 hours and 15 minutes closer to the AIME office than the mine, counting air and automobile travel). Mr. Wheeler was at Chuqui when they turned the first shovelful of earth, has an understandable pride and affection in the mine, and kindly offered to make all arrangements. As a guest of Chilex, physical exam, injections, police clearance, passport, and airline reservations on the crack Panagra airline are procured swiftly and almost painlessly. Winter clothes were advised by Mr. Wheeler and D. M. Dunbar, who is the former's assistant and a seasoned Chuqui-ite. Both of these gentlemen obviously prefer Chuqui climate, with 364 days of sunshine, 1½-in. rainfall, rare atmosphere of 10,000 ft elevation, Christmas-time earthquakes, and howling winds through June and July to the vicissitudes of New York weather.

Panagra's de luxe DC-6 noses in toward land at Antofagasta, Chile, the supply and embarkation port for Chuquicamata. The Coastal Range, completely barren of green, thrusts a spur far out into the Pacific creating a crescent-shaped and protected harbor at Antofagasta. Beach gives way to pampa, to hills, and the only variation in scenery is in topographic relief. Chilex's reception committee in the person of the alert and amiable Bob Condit, assistant port agent, takes care of everything: immigration, interviews by the local press, baggage, transportation, and introduction to Chilean cuisine.

The drive up to Chuqui takes about three hours, on the only two lane highway in northern Chile; hard surfaced, with a few kilometers of repair work in progress. We were fortunate to make the trip with manager Charles Brinckerhoff, who arrived the same day at Antofagasta from the seat of government at Santiago where he goes frequently to keep in touch with the company's Santiago office.

Brinckerhoff looks like a mining engineer, big, sturdy, and of affable personality. He talks over the problems of his staff at all levels, presses for decisions, and is not bound by the book. Trained as a metallurgist at Columbia followed by a year or so in metallurgical work for Phelps Dodge, he moved to Inspiration where he learned mine operation in general and block caving in particular. With over 20 years mining experience, eleven of which were at Potrerillos, as assistant mine superintendent, mine superintendent, and finally as manager of Andes Copper Mining Co., he was a logical choice to take over at Chuqui. During the past four years he has minimized the effects of inflation on production costs at Chuqui.

Supplementing Brinckerhoff is assistant manager, William Wraith, Jr., who came from Cananea and is experienced in metallurgy and management. These two operate as a team, Wraith following metallurgy closely and filling in overall in Brinckerhoff's absence. Bill is an ardent sportsman and conceived the idea shortly after his arrival at Chuqui of stocking trout in near-by streams. Like everything else at Chuqui, the trout are big—catches of 2-year olds weighing 8 lb are not unusual. The fishing provides a welcome diversion in a camp like Chuqui where folks are inclined to eat and sleep mining.

In the mine, Chuqui-trained G. S. Wyman, superintendent, and L. E. Fish, assistant superintendent, have an unusually complex ore handling problem. Four types of rock must be kept separate, oxide ore for the leaching plant, mixed ore stocking for future treatment, sulphide ore for the new plant, and waste rock. We commented on the fact that methods and equipment were right up-to-date. Says Wyman, "we try anything new that comes out of the States, or anywhere else for that matter."

Wrestling with the exigencies of bringing in the new sulphide plant, Edward D. Tierney, also a Chuqui-trained engineer, is apt to wake up at 5 a.m. buzzing with ideas for the day's effort. Big of stature, even the altitude does not stop him from moving swiftly from his snug-fitting swivel chair to the scene of operations. Thoughtfully shifting his chewing gum from one side to the other (he chews gum on the job because cigars "slow him down") Ed says "With men like Sanders and Witcomb in the mill and Morris and Towell at the smelter our trouble spots just naturally get worked out." D. S. (Scotty) Sanders was superintendent of concentration for Cerro de Pasco and wrote the article on the subject in the Cerro de Pasco issue of *Mining and Metallurgy*, November 1945. As superintendent of the mill he is busy trying to get design tonnage through the plant, a common problem when starting up a new mill. Ted Witcomb is his assistant. Ted's father was also an engineer at Chuqui. G. E. Morris, smelter superintendent, and Frederick Towell, assistant, were imported respectively from Potrerillos, Chile, and Miami, Ariz.

Plant design was done by Anaconda's own staff under Wilbur Jurden. This group also designed Morenci and many of the same engineers wrote articles for the Morenci issue of *Mining and Metallurgy*, May 1942. Trouble shooting for the design staff and ensconced for the past two years at the mine is Lester R. MacLeod. He sits opposite Ed Tierney in the office, looks thoughtful, and after a period of such preoccupation will disappear to observe some detail of plant layout. Lester's reputation is based on being able to make things work.

Chuqui by night is a pendant of lights on a gown of blackness. Scones of fire appear at refinery stacks. The plant and community sprawl in a line over a distance of six kilometers. From the camp, the mine is hidden around the corner of a mountain; the sulphide plant is visible as two stacks, the buildings being obscured by the ripio (tailing) piles. The oxide plant is likely to be wreathed in a haze of industrial activity while the old and new camp and staff quarters gleam in the bright sun. New prefab concrete houses have been done in pastels to add color to the community. The new workmen's quarters compare favorably in space and equipment to the Levittown development on Long Island. Of course, television sets were not built into the wall.

One morning after breakfast we were confronted by a tall smiling man. He had a four-wheel-drive jeep, provisions, extra clothing, and an eagerness to be on the way. This person proved to be W. E. Rudolph, chief engineer, pampa rat, member of the Explorers' Club (usually accompanied by Mrs. Rudolph), and raconteur. Our objective, we were told, was a study

of the water supply for the sulphide plant which comes from the Rio Salado (and it is a salt river) dammed by Chuqui engineers some 40 km east. Besides fulfilling our objective, Mr. Rudolph briefed us on the mores of the Indians, pampa lore, and the names and heights of the numerous volcanoes toward which we drove. He discussed small scale sulphur and salt mining and archaeological work of the district, mountain climbing legends, and how to cope with being stranded on the pampa or in the Cordillera.

On the construction of water facilities he was extremely interesting, but one bit of ingenuity particularly struck our fancy. With all the care being exercised to prevent corrosion by erecting a deaerating plant, and keeping foreign matter from the pipe by settling basins, it was necessary not to overlook a peculiarity of Indian workmen. In addition to their many excellent qualities, they are inclined toward pranks which take the form of leaving tools or stones in the pipe. To prevent this an Indian was assigned the task of crawling through each new section of pipe as it was installed. At nightfall he was permitted to come out and the open end of the pipe was securely covered. This gentleman, by the end of construction, had crawled through upwards of 40 km of piping. Such stunts as this have probably given rise to the Chuqui version of the old Army bromide: There are three ways to do things; the right way, the wrong way, and the Chuqui way.

The rigors of foreign mining cause men to grow rapidly to engineering maturity. Such a case was one Less Wheeler. A man in his middle twenties, he was on his first contract doing routine engineering when the Rio Salado water project was about to start. He was considered a bright lad, but inclined to dreaming and rather disinterested in his current work. When the specialist imported from the States for dam construction was unable to take the job, Wheeler got the nod and was packed off to live with the Indians at the scene of operations. It was his job to supervise construction and in a matter of weeks there was a striking change in his personality; he became eager, aggressive, and competent—in short, a good executive. Work had to be completed in the dry season but a surprise heavy snowstorm on the Cordillera caught the job at pouring concrete in the bottom of a tight gorge. A thaw would mean torrential waters, trapped men, and wrecking of forms and superstructure. An elaborate system of alarms was set up to clear personnel if the thaw should come, and work proceeded day and night under lights from trucks which were jacked up from the rear to bring headlight beams to bear on the job. In a week, under terrific nervous strain, the job progressed to a point which pulled the men up to safety.

Some 25 years earlier on a previous contract at Chuqui, Bill Rudolph made a discovery which later became the site for tailing disposal for the sulphide plant. Bill's exploring instincts led him to take his horse over what he thought to be a short cut to Chiu Chiu. He traversed and ultimately became lost in a *salada* formed in the bottom of a tremendous dry lake bed. When he returned to Chuqui in the forties, two possibilities for tailing disposal were under consideration. Both had their drawbacks. Time was growing short when Bill remembered the *salada*. The area was so big that the necessary survey would have taken months had not a remarkably accurate old map come to Bill's attention. With this as a guide, it was possible to restrict survey work to points which would need damming. The result of this circumstance provided a tremendous storage area which would not require movable trestles.

Characteristic of Chuqui-ites is a succinct statement of facts. On class distinction they say: Most pronounced among domestics; No. 2 girl cooks breakfast for employers and for cook who gets up later. On labor: Energetic, proud, capable after training; but not low cost. Including exchange, store privileges where the Company sells food and other domestic supplies at 1932 prices, medical expenses, housing, welfare, and social security, the employees at Chuquicamata are in the best economic position of any in Chile. On climate: It never rains (on our second day in camp, it *did* rain). Mr. Brinckerhoff explained that it was part of the hospitality program. While visiting the nitrate mines later we explained Mr. Brinckerhoff's control of the elements to John Peeples, manager of Anglo-Lautaro Nitrate Corp. He said that in that case he would order an eclipse of the sun on the morrow. He did too, a three-quarters eclipse.

On leaving Chuquicamata our impression was of a modern mining venture, destined to make even greater history through the combined efforts of a competent staff (only a few of whom has it been possible to credit here), competent and discerning management in New York, and with a plant designed by an exceptional group of engineers.

AIME Announces Two New Publications —

Problems of

CLAY AND LATERITE GENESIS

A collection of authoritative articles on the occurrence, identification, and genesis of clays and the hydroxide minerals of aluminum and iron. The scope of the volume ranges from prospecting techniques through interpretation of X-ray and other experimental data. For both practical and research purposes this book is a must for geologists.

Published 1952. 252 pages.

$4.20 to AIME Members
$6.00 to Non-members

GASIFICATION AND

LIQUEFACTION OF COAL

For all fuels technologists, this book appeals as a composite of current world-wide status of the synthesis of liquid fuels from coal. It is a collection of articles presented by authorities in the field from all over the world. Factual data on pilot plant and commercial operation are included as well as reviews of technology and economics.

Published 1952. 240 pages.

$4.20 to AIME Members
$6.00 to Non-members

American Institute of Mining and Metallurgical Engineers

Box BB **29 W. 39 Street** **New York 18, N. Y.**

AIME Annual Meeting To Attract Registrants From All Sections of Western Hemisphere

MORE than 2500 members of the AIME from all sections of the U. S., Canada, Central and South America are expected to attend the 175th General Meeting scheduled for Los Angeles Feb. 16 to 21, 1953. The Southern California Section will be host for the meeting.

It will be the first time the Annual Meeting has been held in Los Angeles. In keeping with the historic nature of the event, headquarters will be the fabulous Statler Hotel, recently opened in downtown Los Angeles. Meeting plans are under the direction of Henry T. Mudd, General Chairman, assisted by members of the Southern California Section, headed by Basil P. Kantzer.

The annual business meeting is scheduled to be held Tuesday, February 17, at the Statler Hotel, at 4 pm.

Minerals Beneficiation Div. has planned a very important program, with 10 technical sessions, the famous Scotch Breakfast, divisional luncheon, and participation in the Mining Branch and All-Institute activities. The MBD business meeting is set for the morning of Feb. 16, 1953.

The technical program will begin on Monday afternoon, Feb. 16, with a session on Crushing and Grinding. Papers to be presented are: *Design Development of Crushing Cavities*, by H. M. Zoerb; *Crushing Plant Dust Control at Ray Mines Division, Kennecott Copper Corp.*, by J. F. Knudsen; *Work Indexes Tabulated (Impact and Compression Data, Rod and Ball Mill Grindabilities)*, by F. C. Bond; and, *Multiple Stage Grinding and Classifying Circuit for Fluorspar*, by D. Spees. Simultaneously, a joint session will be held with the Extractive Metallurgy Div. Five papers covering the subjects of selection, desirable qualifications, and employment of new technical employees will be presented. Such experts as F. R. Morral, Kaiser Aluminum Co.; Joseph Newton, University of Idaho; Frank Wickhorst, Kaiser Enterprises; and H. R. Spedden, Union Carbide & Carbon Co., will discuss these vital topics.

On Tuesday morning, immediately after the Scotch Breakfast, the technical gathering will hear five speakers on the subject of Materials Handling and Automatic Controls. Papers to be heard are: *Moving a 2000-ton Concentrator and Supporting Town, 165 Miles*, by the staff of Sherritt-Gordon Co.; *Rubber Coating of Materials Handling Equipment*, by the Galigher Co.; *Wet Phosphate Rock Storage and Handling*, by G. L. Lyle, Jr.; *Coordinated Process Control*, H. C. Frost; and, *What the Industry Wants in the Way of Automatic Controls.*

Hollywood Bowl, where music under the starry California night is one of the many great attractions California has to offer. The varied highlights of Los Angeles will be at the disposal of Annual Meeting registrants from the centrally located Statler Hotel.

At the same hour, a four-paper session on the application of centrifuging to the metallurgical industry will take place. Speakers will be Charles Ambler, Donald A. Dahlstrom, and members of the staffs of the Bird Machine Co. and the Oliver United Filters, Inc.

On Tuesday afternoon, Feb. 17, the Solids-Fluids Separation conclave will continue with: *The Horizontal Filter as Applied to the Metallurgical Industry; Application of the Dewatering Screen to the Metallurgical Industries;* and, *Magnetic Separators as They Relate to the Concentration of Magnetic Iron Ores.*

Also on Tuesday afternoon there will be a joint session with the Industrial Minerals Div. with papers on beneficiation of the nonmetallics.

Sessions will be held Wednesday, Feb. 18, both morning and afternoon. The programs will be on Concentration and Flotation Theory, with papers on: *Effects of Alkalinity on the Flotation of Lead Minerals*, by M. G. Fleming; *Leach Precipitation Float Process At Anaconda*, and *Sponge Iron at Anaconda*, by F. F. Frick; *General Article on Molybdenum Recovery; Quantitative Bubble Pick-up Methods*, by Shiou-Chuan Sun and R. C. Troxell; *Flotation of Copper Silicate Minerals*, by C. G. DeWitt; *Double Bond Reactivity of Oleic Acid During Flotation*, by A. M. Gaudin and R. E. Cole; and, *Analytical Determination of Technical Xanthates by Potentiometric Titration*, by C. H. DuRietz.

In addition another joint gathering will be held Wednesday afternoon with the EMD in Pyrolysis and Agglomeration. The four articles to be presented are: *Operation of a 12-Ft Sintering Machine at the Blackwell Zinc Co., Blackwell, Okla.; Development of a Continuous Up-Draft Sintering Process; Pelletizing Southern Hematite Ores;* and, *Testing of Sinter.*

Thursday morning will feature an MBD program on mill design and construction with: *Ore Testing to Determine Flowsheet Prior to Plant Design; Feed Preparation and Ore Bedding; Fundamentals of Mill Design; Methods of Estimating Costs of Plant Construction;* and, *Examples of Recent Mill Designs.*

Concluding the activities for the four days of technical programs, will be the luncheon of the Minerals Beneficiation Div. Following the luncheon, a clean-up session will be held for last minute discussions.

The Coal Div. has prepared a six-session program, the theme of which will be Western, with enough diversification to be interesting and enlightening to all delegates. If operating papers are the required features, articles on such subjects as: *Two-Way Conveyor Belt Transmission; Longwall Mining;* or, *Montana and Wyoming Coal Mining;* will do the job. For both mining and beneficiation, *Expansion Program in Mining and Preparation at Sunnyside, Utah Coal Property*, by Wesley Hyatt and R. G. Heers should prove interesting. Other sessions will include *Mine Ventilation, Lubrication of Mine Equipment*, along with a very promising article on *Coals of Mexico* by

Seated at luncheon are the people who are helping to make the 1953 Annual Meeting a successful reality. Clockwise, beginning at head of table: Mrs. R. H. Johns, Chairman, WAIMME Registration; Newell Appleton, Assistant Secretary, AIME; Ian Campbell, Chairman, Banquet; W. H. Geis, Chairman, Reception; A. C. Rubel, Chairman, Welcoming Luncheon; J. H. Pemberton, Chairman, Hotel; Henry Mulryan, Chairman, Field Trips; Blair W. Stewart, Secretary; C. E. Totten, Vice Chairman, Cocktail Party; James D. Hughes, Chairman, Finance; Evan White, Assistant Secretary; R. E. O'Brien, Field Secretary, AIME; John R. West, Vice Chairman, Publicity; Thomas F. Edson, Chairman, Informal Dance; Laurence Chasteen, Chairman, Technical Sessions Arrangements; E. A. Arthur, Vice Chairman, Field Trips; Nicholas A. D'Arcy, Jr., Vice Chairman, Annual Meeting; Frederic Boericke, Vice Chairman, Annual Meeting, and Chairman, Registration; Harold Clark, Chairman, WAIMME Liaison; Mrs. Harold Clark, General Chairman, WAIMME Annual Meeting; and Henry T. Mudd, General Chairman, 1953 Annual Meeting.

Cortes Obregon of the Bank of Mexico. L. A. Turnbull of the U. S. Bureau of Mines will discuss the methods and results of *Eliminating Methane from Coal Seams Before Mining*. The principal papers covering important phases of coal beneficiation are: *Drying Fine Sizes of Coal*, by G. A. Vissac; *Heavy Medium Coal Cleaning*, by H. F. Yancey and M. R. Geer; and, *Reduction of Moisture Content of Washed Coals*, by Orville Lyons.

The Industrial Minerals Div. has forged ahead in formulating a program for the February Annual Meeting in Los Angeles.

For the Cement, Lime and Gypsum sessions, papers on *Gypsum in the Aircraft Industry*, *Geology of San Marcos Island Gypsum Deposit*, and *Status of New Dry Rotary Blast Hole Drills*, will constitute the program.

The Chemical Raw Materials sessions will feature a symposium on potash, with 13 papers to be presented. Airborne and Ground Magnetometer Work has been suggested. Four papers, *Mining Phosphate Rock*, *Idaho Phosphorous*, and *Geology of Carlsbad Potash* and *Geology of Pearlite* will be on the Mining Geology program.

Sessions on Dimension stone and slate have been tentatively worked out with papers on *Geologic Factors Affecting the Dimension Stone Industry of California*, *A Quarter Century of Granite Quarrying in Massachusetts*, *Rock Bursting and Other Geologic Problems in the Granite Quarries of Barre, Vt.*, *Quarry Practices in the Indiana Building Stone Industry*, *Stratigraphy and Composition of the Salem Limestone*, and *Dimension Stone in the U. S. in 1952*.

Rare Minerals Session (Joint Session with the Society of Economic Geologists)

1. *New Technique for Grading Corundum Ores*, by H. F. Carl, A. Hockman, and H. W. Jaffe
2. *The Synthesis of Industrial Gems*, by Clifford Frondel
3. *Monazite Deposits of the California Piedmont*, by J. B. Mertie
4. *Geochemical Association of Columbium and its Geological and Economic Aspects*, by M. Fleischer
5. *Minor Element Accumulation in Coal Ash and its Economic Implication*, by T. Stadnichenko

Plans for four Geology Subdivision sessions are complete, with the theme of the program to be the application of geology to exploration, development, evaluation, and appraisal. The papers scheduled for these sessions include: *Exploration of the Blackbird Cobalt District of Idaho* by John S. Vhay; *Development of the Iron Ore Deposits in Labrador* by J. K. Gustafson; and *Critical Factors in Modern Appraisal of Mineral Deposits* by Harrison Schmitt, Geologist, Silver City, N. Mex.

Society of Economic Geologists Sessions will feature an important session on the *Nature and Origin of Ore-Forming Fluids* in the form of panel discussion. Another session will be on *Mineral Deposits of The Southwest*, a symposium.

The Mining Subdivision program will be a substantial one, with an impressive list of technical papers on tap. Among the scheduled papers are: *Development of St. Joseph Lead Co.'s Indian Creek Property*, by C. Kremer Bain; *Mechanization at Fredericktown-National Lead Co.*, by Harold Krueger; *Silver Bell Mines*, by Alvin A. Smith; *Block Caving to the Bottom of an Open Pit at Inspiration*, by H. Carroll Weed; *Block Caving at Bunker Hill and Sullivan*, by Charles Schwab; *Resurrection Mining Co. and Modified Top Slicing Adaptation*, by John H. Wright; and *Stripping a Small Orebody Preparatory to Mining*, by P. W. Lewis.

Iron Mining sessions will have *Eagle Mountain Iron Ore Operation*, by K. B. Powell; *Warren Pipe & Foundry Co., Mt. Hope Mine, Mt. Hope, N. J.*; and *Iron Ore Production at the Cornwall Mine, Bethlehem Steel Co., Cornwall, Pa.* by S. J. Shale.

International Salt Co. Room and Pillar at the Heath Mine, Plasterco Va., will be a feature of the Non-Metallic sessions.

Joint Mining and Geology sessions will hear Peter Joralemon on *Geologists vs. Gehooligists*, and Don Yardley on the *Sun Dial Compass*.

The Geophysics Subdivision plans a varied program, with mining aspects of geophysics the central topic of papers to be presented. Featuring the sessions will be a symposium on geochemical methods. It will mark the first time that geochemistry as applied to prospecting will make an appearance at the Annual Meeting.

Subjects already scheduled for the program are gravimetric, airborne magnetometer, and ground magnetometer methods. Another showing has been proposed for the USGS movie *Airborne Magnetometer*, acclaimed at the last Annual Meeting and the Chicago Fall Meeting.

The welcoming luncheon is slated for Monday noon, with a national figure as the principal speaker. The pre-smoker cocktail party and dinner-smoker will be enlivened by some of the outstanding theatrical, radio, and television stars on the Coast. Divisional dinners on Tuesday evening will prelude the informal dance. The President's reception and Annual Banquet is scheduled for Wednesday evening. Division luncheons will be held Tuesday and Wednesday noons.

Wives of AIME members have not been forgotten, with a round of fashion shows, luncheons, and tours through Hollywood studios on the list of activities.

Assisting Henry T. Mudd are the following committee Chairmen: A. C. Rubel, welcoming luncheon; Samuel Bowlby, cocktail party; Ken Powell, dinner smoker; Thomas Edson, informal dance; Ian Campbell, banquet; James D. Hughes, Finance; William Geis, reception; Fred Boericke, registration; L. Chasteen, technical sessions; Henry Mulryan, field trips; Mrs. Harold Clark, ladies; C. W. Six, treasurer; John R. West, publicity; Paul Andrews, student activities; and John Pemberton, hotel.

Magnetic Roasting of Lean Ores

by Fred D. DeVaney

DURING the past few years a radically new process for the magnetic roasting of iron ores has been investigated and developed by Pickands Mather & Co. and the Erie Mining Co. in the Erie laboratory at Hibbing, Minn. This process, originally devised by Dr. P. H. Royster of Washington, D. C., involves the use of a roasting technique quite different from older methods. It has now been demonstrated that iron-bearing materials can be roasted as effectively as by any previously known method, and at a much lower cost.

The increasing shortage of highgrade iron ores in this country has accelerated the search for new methods that would permit low grade materials to be utilized. The concept of magnetically roasting low grade nonmagnetic ores such as the oxidized taconites and then separating such material magnetically has always had considerable appeal. The magnetic concentration idea is attractive because of the sharpness of the separations and cheapness of the method. Heretofore, however, the equipment and the processes available for the magnetizing-roasting step have left much to be desired. The customary equipment available for reduction roasting has been: 1—multiple hearth furnaces, 2—rotary kilns, and 3—shaft type kilns. In addition, it is understood that some work has been done in magnetically roasting fine ores by a process using the FluoSolids principle, but little information on this process is available. The multiple hearth kiln has been used the most but first costs and operating costs have been high because of low capacity, high maintenance, and poor gas utilization. Magnetic roasting can be done in a rotary kiln, but the radiation losses are high and the conversion to magnetite is usually unsatisfactory because of poor contact between the gases and the solids. Of the shaft-type furnaces, probably the most efficient yet developed is that designed by E. W. Davis of the Minnesota Mines Experiment Station. This furnace was operated at Cooley, Minn., during 1934-1937 but was abandoned in 1937 because the operation was uneconomic.

Heretofore the basic concept behind most magnetic roasting processes has been the idea of heating iron ore to a temperature of 800° to 1100°F in a strong reducing atmosphere, preferably either carbon monoxide or hydrogen. Temperatures under 800°F were undesirable since excessive roasting time was required. Temperatures over 1100°F were avoided because of the danger of converting part of the iron to ferrous oxide which is nonmagnetic.

In the new roasting process, the operation is carried on in a shaft furnace using a controlled atmosphere containing a low percentage of reducing gas. The temperature in the roasting zone is considerably higher than with the usual reducing gas and this speeds up the reduction time. Portions of the spent furnace gases are cooled and recirculated and this together with the good contact between ore and gas makes for high reducing gas utilization. High heat economy is secured by recuperating heat from the roasted ore by passing the cold reducing gases countercurrent to flow of ore. The heat transfer principle is similar to that employed in a pebble stove and to that used in the Erie Mining Co. furnace at Aurora, Minn., for pelletizing fine magnetite concentrates derived from taconite.

The theory of controlled atmosphere during the roasting operation can best be appreciated by inspecting the equilibrium diagram of the Fe-C-O system shown in Fig. 1. An inspection of this diagram shows that in certain areas magnetite, Fe_3O_4, is the only stable form of iron. A further inspection of this table shows that if the proper ratio is maintained between carbon dioxide to carbon monoxide, such a gas will be reducing with respect to hematite, Fe_2O_3, and will be oxidizing with respect to both ferrous oxide, FeO, and iron, Fe. It should be kept in mind that the formation of ferrous oxide in a roasting operation is harmful, since this oxide is nonmagnetic; if it forms in any quantity, it will cause substantial loss of iron in the ensuing magnetic separation step. If a ratio of approximately three parts carbon dioxide to one of carbon monoxide is maintained, the resulting operation can be carried on at a relatively high temperature without fear of over-reduction. Specifically, most of the tests in the Erie furnace have been made at a temperature of 1500° to 1600°F, with an entrant gas containing approximately 5 pct carbon monoxide and 15 pct carbon dioxide, with the remainder largely nitrogen. It should be remembered that the ratios of carbon monoxide to carbon dioxide shown in Fig. 1 hold even though the bulk of the gas is an inert gas such as nitrogen. It may surprise many to learn that a gas containing as low as 3 pct carbon monoxide, and 12 pct carbon dioxide with the remainder nitrogen is an extremely effective reducing gas in the 1000° to 1600°F temperature range. The reducing gas is not limited to carbon monoxide, and mixtures of hydrogen and carbon monoxide may be used effectively, provided that a similar ratio is maintained between the reducing gases and carbon dioxide and water vapor. For a more detailed explanation of the theory involved, the reader is referred to U. S. patents 2,528,552 and 2,528,553.

From a safety standpoint, the weak reducing gas used in the furnace offers an advantage. Its composition is such that it is well below the limits of explosion should air enter a hot furnace. This condition is not true with the usual reducing furnace, in which a gas rich in carbon monoxide or hydrogen is used.

The general furnace design and method of operation may best be understood by an inspection of

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Figs. 2 and 3. Fig. 2 shows a cross-section of the magnetic roasting furnace system at Hibbing. This furnace is circular and has an ID of 2½ ft. It is capable of roasting from 1 to 1¼ long tons of iron ore per hr. In practice, the furnace is charged with crushed ore and is fed through either a double bell arrangement such as is used on blast furnaces or by the hopper-like device shown. The shaft of the furnace is filled completely with ore, which moves down the shaft as the material is discharged from the bottom. Basically, the furnace consists of an upper and lower stove with a middle chamber in between. A mixing chamber and combustion chamber are also provided. An optional furnace design is to have two stoves in parallel and operate them intermittently, as in the manner of a pebble heater. Since two stoves are superimposed one upon the other in the present installation, it is called a tandem furnace. In the upper stove, the cold moist ore is heated up to the desired temperature by uprising gases. The gases are of such composition, to be discussed later, that the descending ore is reduced and the iron oxides converted to magnetite. Ore from the upper chamber travels through the middle zone of the furnace where the area of discharge into the lower stove may be restricted. Weak reducing gas is blown up through the lower stove, cooling the hot descending roasted ore. The purpose of the middle stove is to make it easier for gases from the lower stove to pass through the combustion chamber than to travel up through the central part of the shaft. Because of the lesser resistance to the flow of gas to and through the combustion chamber, a greater portion of the gas follows such a path, and in the mixing chamber these gases are brought up to the desired temperature by a mixture of gases from the combustion chamber. A separate combustion chamber is provided so that the fuel added can be burned completely with little or no excess of oxygen. It is very desirable that the amount of air to be used for combustion be held to a minimum since an excess would tend to cause combustion of the gases used for reduction. As usually operated, the temperatures of the gas leaving the mixing chamber are held at 1500° to 1600°F. This heated gas passes up through the ore and comes off the top of the furnace at a temperature of 180° to 300°F. The ore is discharged from the bottom of the furnace at a temperature of 150° to 250°F. With some ores, it is desirable to water-quench this material to prevent oxidation.

The gases blown through the furnace serve a two-fold purpose, to reduce the iron oxides and also to act as a carrier gas or heat exchange media. For a carrier gas, the amounts must be closely regulated so that sufficient cooling gas is supplied at the bottom of the furnace to abstract the heat from the discharged ore. Likewise, the volume of heated gas must be such that it supplies the necessary heat to the raw ore fed to the furnace. This amount of carrier gas can be calculated readily from the specific heats of the gases and ore, and in operation the amount so blown conforms closely to the theoretical. The amount of gas blown is usually from 10 to 12 cu ft per lb of ore roasted. To obtain a suitable amount of carrier gas of the proper analysis and to secure reducing gas efficiency, a major portion of the exit gases from the top of the furnace is scrubbed and cooled and then recirculated. The amount of gas wasted from the top of the furnace is equal only to the products of combustion from burning the oil or gas necessary to supply heat for the process in addition to water vapor coming from the ore. In the Hibbing furnace, reducing gas is produced in a slagging-type gas producer. This gas has a composition of approximately 34 pct carbon monoxide, 1.8 pct hydrogen, 0.6 pct carbon dioxide, 0.4 pct water, and 62 pct nitrogen. It has been found that an efficient roasting procedure can be achieved if the carrier gas entering the bottom of the furnace contains 6 pct carbon monoxide. This carrier gas is made up of the spent exit gas from the top of the furnace, which will normally carry from 1½ to 3 pct carbon monoxide and an amount of new gas from the gas producer to bring the average to 6 pct carbon monoxide. The composition of this gas is easily set and maintained by means of a combustible gas recorder and by instrument-actuated valves. The combustible gas indicator analyzes the gases continuously, and through interlocking controls varies the amount of rich gas admitted to the system to hold the gas composition constant.

The basic reducing reaction that takes place in the roasting zone is: $3Fe_2O_3 + CO$ to $2Fe_3O_4 + CO_2$.

This reaction is exothermic and liberates 88 Btu's per pound of magnetite formed. This heat is liberated in the upper stove and contributes substantially to the heat economy of the system. The mass of ore in the constricted area in the middle stove serves a useful purpose other than a heat recuperator in that it insures that a greater portion of the gas from the lower stove will pass through the mixing chamber and be brought up to the desired temperature. This permits a close control of the temperatures in the roasting zone. The control temperature of the operation is the temperature of the gases leaving the mixing chamber. The maximum temperature in the furnace, however, is in the upper stove about a foot down from the stockline where much of the heat of reaction is liberated. The peak temperature at this point is usually 150°F higher than the control temperature. This peak temperature can be indirectly controlled by adjusting the control temperature in the mixing chamber by varying the amount of fuel burned. By use of the carrier gas principle and the burning of fuel in a combustion chamber, the temperature in any part of the furnace can be held constant within 20°F. This insures optimum roasting conditions.

Fig. 3 is a schematic drawing of the furnace and of the auxiliary equipment. The gas at Hibbing is

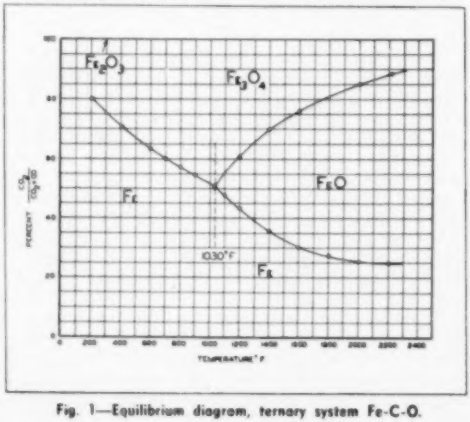

Fig. 1—Equilibrium diagram, ternary system Fe-C-O.

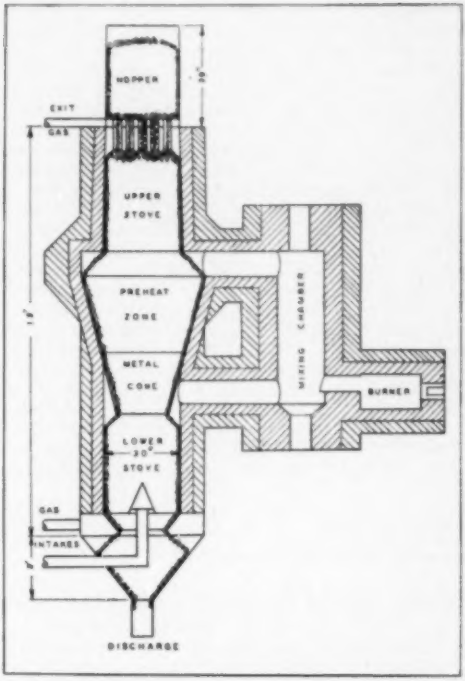

Fig. 2—Tandem magnetic roasting system.

supplied by a slagging-type gas producer, D. This gas, after being cooled and scrubbed, C, joins with a portion of the exit gases from the furnace and is compressed in a blower, E, and blown into the bottom of the furnace, A. The amount blown is adjusted to the feed rate to the furnace. The heat to the system is supplied by burning oil or gas in a Dutch oven, H. Air for the combustion of this fuel is furnished in stoichiometric quantity by blower, G. An amount of exit gas is vented equal to the volume of gas formed by burning fuel and water vapor from the ore. The entire system is under pressure and the amount vented takes care of itself automatically if the vent is made small enough so that there is a positive pressure at this point. The remainder of the top gas is scrubbed and cooled in the scrubber, C, and then joins gas from the gas producer.

During the past five years, in the development of the process, many low grade iron ores from the Lake Superior district have been roasted. Such materials as oxidized taconite and low grade ores from the Minnesota, Wisconsin, and Michigan ranges have been tested. In general, the optimum roasting technique, as far as temperature of roast and gas composition are concerned, is similar for all. However, structural differences such as density, porosity, and distribution of fines have a great effect on the roasting operation.

For purposes of illustration, a typical test on an oxidized Minnesota taconite will be reported in detail. The ore used was a banded oxidized taconite from the lower cherty horizon in the Hibbing area. The iron content was 29.64 pct. Hematite was the predominant iron mineral. The amount of magnetite in the crude ore was small, and the recovery by magnetic concentration alone on a finely ground sample was less than 3 pct. The ore was crushed to pass a 1-in. sq opening, and this was fed to the furnace without screening. Screening at 8 mesh is required on ores that contain many fines, but many of the oxidized taconites can be crushed and few fines formed. The rate of feed to the furnace was 1 long ton per hr on a dry basis. The feed rate is set by adjusting the variable discharge mechanism at the bottom of the furnace to discharge at a given rate and by keeping the hopper at the top of the furnace full. For this test, the amount of carbon monoxide in the entrant gas was set at 6 pct. This setting is made on the Bailey combustible meter, and the meter continuously analyzes the gas going to the bottom of the furnace and actuates a valve from the new gas supply so that the gas composition is kept constant. The same meter continuously measures and records the amount of combustibles in the gases at the top of the furnace.

The amount of carrier gas blown is calculated from the feed rate, which was 2240 lb per hr, or 37.4 lb per min. The amount of carrier gas required was from 10 to 12 cu ft per lb of ore. The initial amount of gas blown was 410 cu ft per min, but it was found that to maintain the best operating conditions it was necessary to increase the gas to 450 cu ft per min, or 12 cu ft per lb of ore. The volume is varied by adjusting a valve between the blower, E, in Fig. 3 and the bottom of the furnace. The rate of blow determines the position of the heat line in the upper stove and also the temperature of the discharged ore. It should be remembered that the heat line in the upper stove is determined by the rate at which the ore descends and the heat wave rises. When in balance, the heat line is static. If the rate of blow is excessive, the heat line will be close to the stock line, the exit gas will be hot, and heat will be wasted in the exit gases. If the rate of blow is too low, there will be an inadequate quantity of carrier gas to take the heat from the ore being discharged and heat will be lost in the ore discharge. This is also undesirable because, if ore is discharged too hot, oxidation of the roasted ore may occur. The

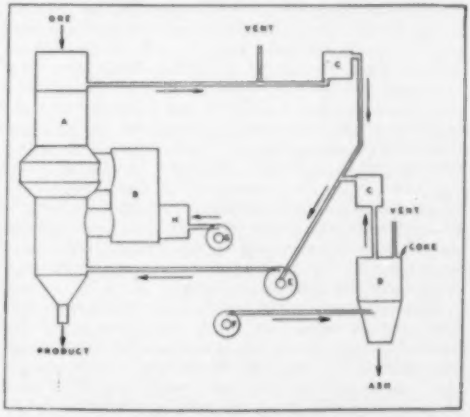

Fig. 3—Ore roasting circuit. The key letters on the figure refer to the following:

A.	Tandem furnace.	E.	Multistage blower.
B.	Mixing chamber.	F.	Positive displacement blower.
C.	Gas scrubbers.	G.	Positive displacement blower.
D.	Gas producer.	H.	Combustion chamber.

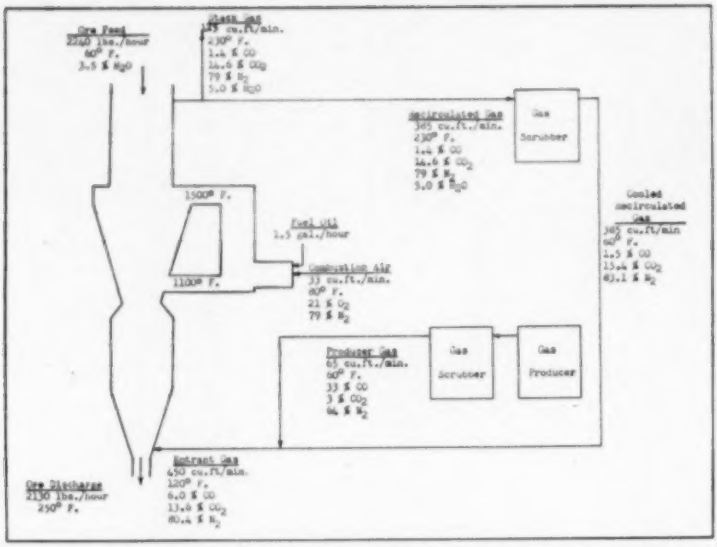

Fig. 4—Diagram showing gas flow, temperature, and composition. Note: All volumes at standard conditions.

position of the heat line is readily controlled manually by varying the rate of blow so that the temperature of the exit gases at the top furnace is held at an optimum. This can also be controlled closely by instruments which regulate the rate of blow so that the exit gases are held within narrow temperature limits. In the example cited, the temperature of the gases leaving the mixing chamber was held at 1500°F. This heat was secured by burning oil with closely controlled quantities of air in a small combustion chamber. After the furnace had reached a state of balance, the amount of oil required was found to be 1.50 gal per hr, or 1.50 gal per long ton of ore roasted. The amount of carbon monoxide required to reduce a pound of iron in the form of hematite to magnetite is 1.12 cu ft. In the example given, 65 cfm of producer gas containing 33 pct carbon monoxide was used per min. Of this amount 8.4 pct was wasted with the exit gases, 59 pct was used to reduce the ore, and because of air leakages 32.6 pct of the gas was burned in the furnace. The gas burned, which has a heating value of 100 Btu per cu ft, contributed a total of 126,600 Btu per hr to the system. The total heat supplied to the roasting operation, from the burning of oil and a portion of the reducing gas, was 327,000 Btu per hr or 327,000 Btu per long ton of ore roasted. In a properly designed large furnace with good insulation the heat requirements for roasting a similar ore should not exceed 200,000 Btu per long ton.

This low fuel consumption is made possible because of the excellent recovery of heat from the roasted ore by the carrier gases coming up through the lower stove. An 1100°F preheat of this carrier gas is secured, and the only additional heat that need be supplied is an increase of temperature to 1500°F. A peak temperature of 1637°F was reached about 18 in. down from the stock line in the upper stove where the exothermic reaction of hematite to magnetite takes place.

Most of the pertinent data for this test, relative to feed rates, gas volumes, gas analyses, and furnace temperatures, are shown graphically in Fig. 4.

After the ore was roasted, magnetic concentration tests were made on ore ground to 150 mesh. This fine grinding was required on this particular ore to give sufficient liberation to make acceptable grade concentrates. These results are shown in Table I, where it may be noted that a weight recovery of 43.77 pct was made of a concentrate having a grade of 63.25 pct iron and 7.56 pct silica.

The overall iron unit recovery was 90.63 pct. It should be remarked in passing that the magnetic roasting step, in addition to putting the ore into a form that can be readily and cheaply concentrated, usually makes it much more amenable to grinding. The power required to grind a roasted taconite is about half that needed to grind unroasted taconite.

The physical characteristic of the ore to be roasted greatly affects its preparation for roasting. Some porous ores need be crushed to only 2 in. and a good roast can be secured. Other dense ores require crushing to ⅜ in. to secure good penetration of the reducing gas, or the roasting time must be increased. In general, a decrease in feed size is more effective than an increase in roasting time. The amount of fines in the ore to be roasted is very important, as the fines greatly affect the resistance to the flow of gas. An increase in fines increases the pressure the furnace blower must operate against, and thereby increases the horsepower requirements. The

Table I. Magnetic Concentration Test on Roasted Taconite Ground to 150 Mesh

Product	Wt, Pct	Iron	Silica	Total Iron, Pct
Crude ore		29.64		
Roasted ore	100.00	30.54		100.00
Magnetic concentrate	43.77	63.25	7.56	90.63
Nonmagnetic tailing	56.23	5.08		9.37

amount of −10 mesh material that can be tolerated is about 20 pct. Without question, the furnace works best on a charge containing no −10 mesh material. With such a charge, the power requirements are minimum and heat transfer is more uniform.

With unscreened material, dust collectors must be provided in the flues between the lower stove and the mixing chamber, or at the bottom of the mixing chamber itself. The temperature of both these points is well below the fusion point of any of the ore constituents, so that there are no slagging problems.

With most ores, the solution of the fines problem is to screen and then briquette the fines with a cheap binder, such as waste sulphite liquor, and charge these briquettes into the furnace with the coarse material. Such briquettes hold together well during the roasting operation and are sufficiently porous so that the conversion to magnetite is excellent. An alternative method applicable to some ores is simply to roll the fines in a balling drum and charge the balled fines into the furnace with coarse material.

The furnace may be used for purposes other than the magnetic roasting of iron ores. It may be used for reduction roasting of nickel, manganese, and other ores by varying the temperature of the roast and the gas composition to suit the particular application. This has been checked on several ores.

The capacity of the furnace on iron ores is from 5 to 6 long tons per sq ft of shaft area per 24-hr day. The mechanical maintenance on the furnace is low, as there are no moving parts in the furnace. The feeder and the discharge mechanism are both outside the furnace. Radiation losses can be held to a minimum because of the ease with which a shaft furnace can be insulated. The utilization of reducing gases and roasting efficiency both are excellent. Labor requirements are small, as feeding and discharging the furnace is done mechanically. Metallurgical control is taken care of by instruments which hold the temperature of the furnace, the rate of blow, and the composition of the gases within close limits. In conclusion, it may be said that the tandem furnace offers an entirely new approach to the reduction roasting of ores, particularly iron ores, and the results of extended tests on a 1 ton per hr furnace justify the statement that with this furnace reducing roasting can now be done cheaply and effectively.

The Effect of Heat Treatment and Certain Additives On the Strength of Fired Magnetite Pellets

by Strathmore R. B. Cooke and William F. Stowasser, Jr.

The most important factor responsible for the development of strength in fired magnetite pellets containing no additive is the oxidation of the magnetite to hematite and the subsequent recrystallization and grain-growth of this constituent. In the laboratory, pellets having the highest strength and the greatest uniformity of structure were obtained by oxidation at a relatively low temperature and firing between 1300° and 1400°C. Effects of a number of inorganic additives are also reported.

RESULTS presented in a previous paper[1] have shown that the strength of fired pellets made from eastern Mesabi magnetite concentrates containing 8 to 9 pct SiO_2 as quartz and silicates is due to several processes which occur during firing. These are: 1—recrystallization and cementation of the fragmented magnetite grains; 2—formation of slag from reaction between the quartz, silicates, and available ferrous iron, followed by slag-bonding of the magnetite grains; and 3—oxidation of the magnetite to hematite and subsequent recrystallization and grain growth of this constituent.

Microscopic examination of all magnetite pellets fired in one operation in small and large-scale pilot plant furnaces and possessing normal strengths shows that the processes 1, 2, and 3 have taken place; where strengths are low and the pellets are otherwise subnormal, one or more of the processes has not occurred. For example, soft and friable red pellets may contain a central core of slag-bonded recrystallized magnetite which is strong enough in itself, but the shell contains only slightly recrystallized hematite, indicating that the maximum temperature reached was too low.

Processes 1 and 2 can occur only under reducing conditions, as for example, when coal is intentionally added to the original pellets; under neutral conditions, when pellets are fired in an inert atmosphere; or when pellets are brought so rapidly to temperature in an oxidizing atmosphere that oxygen diffusion to the core lags behind the recrystallization and slagging effects. This last-mentioned process has occurred in all pilot-plant shaft fur-

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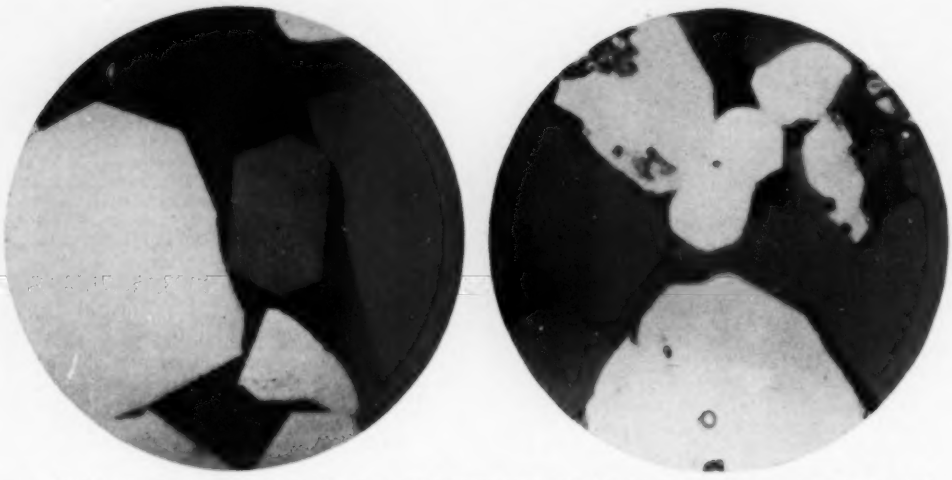


Fig. 1 (Left)—Idiomorphic grains of magnetite (grey) and subhedral hematite in a pellet containing added lime and fired at 1400°C. X1770. Fig. 2 (Right)—Partial conversion of hematite to magnetite in a pre-oxidized pellet fired at 1400°C. X1770.

nace products which the writers have examined, even though oxidation during the heat-exchanging operation subsequent to firing may have removed all but insignificant quantities of magnetite. Hematite concentrates which have been balled with an addition of coal and fired in an oxidizing atmosphere cannot be differentiated structurally or mineralogically from similarly fired magnetite pellets containing no coal.

Factors Responsible for the Development of Strength in Fired Magnetite Pellets

It seems easy to account for the cold strength of a fired pellet when slag has been formed. Prior to firing, a pellet consists of a heterogeneous mass of pores, magnetite, and quartz bonded by relatively weak surface forces. In a substantially neutral atmosphere and at sufficiently high temperature, silicate slag forms in which the magnetite seems to have appreciable but undetermined solubility. The slag wets the solid magnetite and upon cooling functions as a reasonably strong adhesive between the magnetite grains. Strength is further increased by the consolidation of the interstitial pores into a few large holes, so that a much greater surface of the solid constituents is in contact.

On the other hand, it is more difficult to explain the apparently beneficial effect that recrystallization and grain growth of the iron oxides contribute to pellet strength. With both magnetite and hematite there appears to be very considerable mobility at moderate temperatures. In the case of magnetite and just below temperatures at which slag begins to form, small grains are absorbed into larger ones, and a network of this constituent readily develops throughout the pellet. There is a concomitant increase in pellet strength. At higher temperatures growth of the magnetite is accelerated, and provided that there is an appreciable amount of slag, idiomorphic crystals of magnetite appear, see Fig. 1. With only small amounts of slag the crystals are subhedral to anhedral, with much greater areas of contact between neighboring grains. Accompanying the development of idiomorphic magnetite there is a marked decrease in strength of the pellet. It seems that magnetite-to-magnetite bonding is stronger than magnetite-to-slag bonding, and that slag is the weakest constituent developed in pellets. This statement should not be interpreted to mean that slag formation is always undesirable, for the cold strength of a slag-bonded pellet can be higher than that of one in which slag bonding and recrystallization of the metallic oxides is absent. Furthermore, the strength of slag should be increased by the keying action of numbers of idiomorphic grains of magnetite.

Process 3 occurs in normal pelletizing sequence, and cognizant of its importance in heat economy, the writers are also beginning to realize its physical importance in the production of pellets. First, oxidation of magnetite commences below 600°C, so that a relatively tough shell is formed on all pellets before much load comes on to the bed, and deformation of the pellets and consequent extrusion of the viscous cores is prevented when slagging temperatures are reached. Secondly, the freshly-formed hematite does not combine or slag with silica or silicates, so that there is no deformation of pellet shape even at relatively high temperatures, and adhesion between adjacent pellets is normally nonexistent. Above 1300°C, however, significant amounts of hematite transform to magnetite, see Fig. 2, permitting formation of a sticky fayalite slag or of an eutectic between magnetite, silica, and silicates. It is probable that some of the not infrequent formation of lumps of agglomerated pellets in zones of localized and excessive temperature in pelletizing furnaces is due to this thermal decomposition of hematite, although much is due to the extrusion of liquid matrix from broken balls. Thirdly, the higher the firing temperature, the coarser the grain size of the hematite. During the oxidation of magnetite pellets the hematite is formed from the magnetite by a process of grain refinement. There is no physical disintegration, but as certain crystallographic directions within any one magnetite crystal are preferentially oxidized, a single crystal

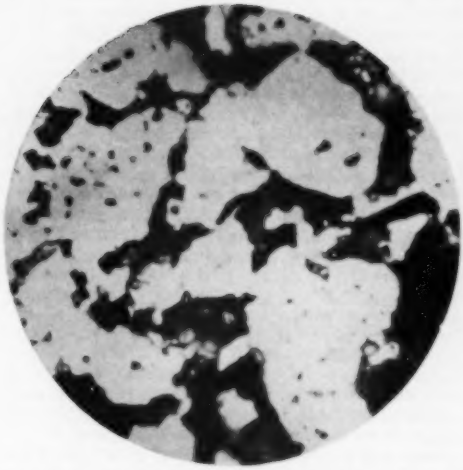
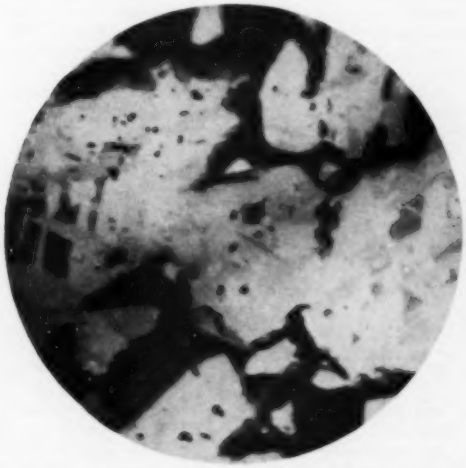


Fig. 3 (Left)—Disappearance of fine hematite to form a network of equiaxed grains. Pellet fired at 1200°C for 30 min in air. X750.
Fig. 4 (Right)—Portion of a large magnetite grain (grey) and idiomorphic hematite. This pellet contained added lime and was fired at 1300°C after oxidizing at 800°C. X1770.

of magnetite eventually becomes a number of independently oriented grains of hematite.

At temperatures around 500° or 600°C the resulting structure is relatively stable, and prolonged heating does not change the orientation or size of the hematite grains. At higher temperatures, however, the hematite grains commence to grow, and what was originally a single crystal of magnetite becomes a single crystal of hematite. Just as grain growth occurs within a grain of this type, so also does grain growth occur throughout the mass of the pellet, the small grains being absorbed by the larger, until a network of equiaxed hematite grains is established throughout the pellet, see Fig. 3. With the coarsening of the grain size, there is a concomitant increase in strength. Because no slag is formed below 1300°C, there can be no transfer of iron atoms through a second constituent, and hematite grain growth below that temperature is dependent only upon the mobility of the iron atoms in that mineral. That the mobility is a real phenomenon is substantiated by carefully controlled laboratory experiments in which magnetite pellets have developed an outer coating of needle-like and radially directed hematite crystals between 1 and 2 mm in length.

The structure developed within the pellet by these mechanisms is analogous to that developed by magnetite in a neutral atmosphere, and the strength of the pellet increases greatly to a maximum at about 1300°C. Between 1300° and 1400°C, however, the quartz and silicate grains which have hitherto preserved their individuality disappear and form slag.



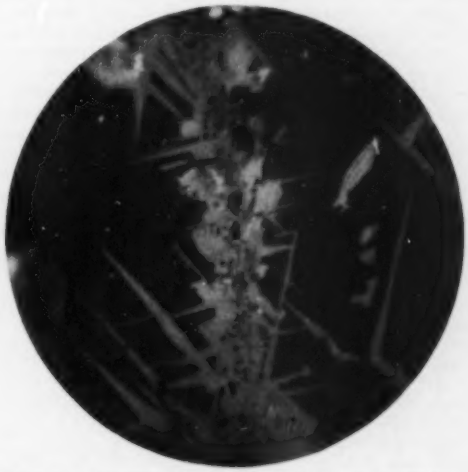


Fig. 5 (Left)—Cementation of three neighboring grains obtained by heating a magnetite pellet to 800°C in air. The same phenomenon, although less pronounced, can be observed in pellets oxidized at 600°C. X2000. Fig. 6 (Right)—Line of cementation between two magnetite blocks clamped together, heated to 800°C in air for 2 hr, and subsequently furnace-cooled. X2000.

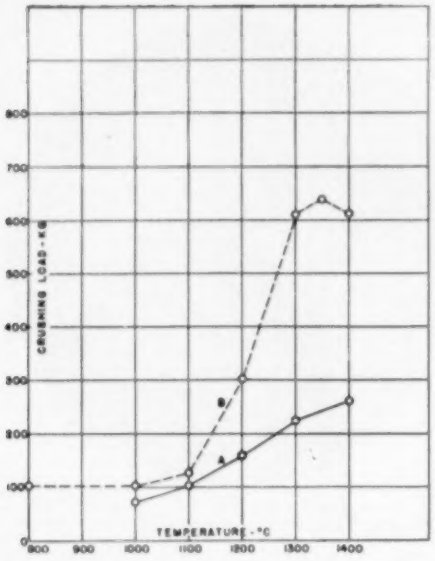


Fig. 7—Cold strength vs firing temperatures for magnetite pellets without additives. Curve A represents strength of single fired pellets and Curve B strength of pellets pre-fired at 800°C in air, 2 hr, and refired at temperatures indicated.

At the same time the continuous hematite network breaks up, and large, idiomorphic, and disconnected crystals of hematite and magnetite appear, see Fig. 4. A decrease in pellet strength accompanies this change. The strongest pellets made from magnetite are those in which there is a continuous network of coarsely crystalline hematite in a matrix of unaltered quartz grains, with a very small quantity of interstitial slag interlocked with the hematite.

Previous work[1] has shown that at firing temperatures between 300° and 1000°C magnetite pellets, compared with identical pellets fired in a neutral atmosphere, are superior in strength if they are fired in air. The strength appears to be due to cementation of contiguous oxide grains at the instant of oxidation, the iron atoms at that time having such high mobility during the lattice transformation that free exchange and rearrangement is possible. Once cementation is effected, further transfer occurs in the normal process of grain growth, the larger areas of crystal contact giving an overall increase in pellet strength. Microscopically, this initial cementation can be observed at temperatures as low as 600°C, see Fig. 5. It will be noted from the photomicrograph that the orientation of adjacent grains which have bridged is different, thus removing the condition of like crystallographic orientation of the original magnetite grains as a requirement for cementation.

A simple experiment illustrates this cementation during oxidation. Two small blocks of coarsely crystalline magnetite were cut and ground to flat surfaces. The blocks were placed with their plane surfaces in contact and held together under considerable pressure with a screw clamp, the area of surface contact being approximately ½ sq in. The assembly was heated in air for 2 hr at 800°C and then furnace-cooled. The magnetite blocks adhered to each other when removed from the clamp and were sectioned across their plane of contact and polished. Fig. 6 shows the contact area. Obviously air has had access to the faces in contact, but oxidation within the mass of the blocks was minor. It is noteworthy that hematite blocks and crystals subjected to the same treatment do not cement together, and likewise individual grains of magnetite resting loosely in contact with each other do not cement when heated to the same temperature. From these experiments it is concluded, first, that oxidation of magnetite to hematite results in a temporary state of disorganization with accompanying enhanced mobility of iron atoms so that particle-to-particle bridging occurs, and, secondly, that high unit pressures are required to make the bridging process effective.

There is a further and most important gain from oxidizing the pellet throughout at moderate temperatures, at least to the extent of forming a superficial layer of hematite on all magnetite grains, before firing at high temperatures. The pellets resulting from this double firing procedure are very homogeneous, for on testing for strength they break in two to four pieces without the production of fines. In contrast, a pellet fired directly at high temperature and without pre-oxidation possesses a duplex structure in which the core has shrunk from the shell, and fractures, upon breaking, into pieces and fines.

In the earlier work by Cooke and Ban[1] eastern Mesabi magnetite pellets approximately ¾ in. in

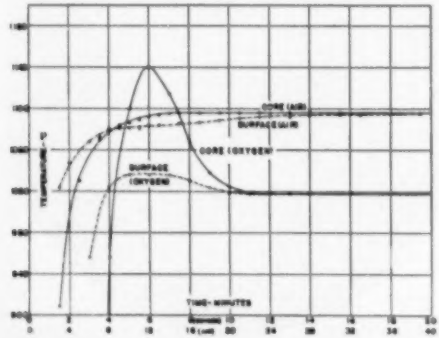


Fig. 8—Temperature differential between surface and core of eastern Mesabi magnetite concentrate pellets fired in air, 5 cu cm per min per g, and in oxygen.

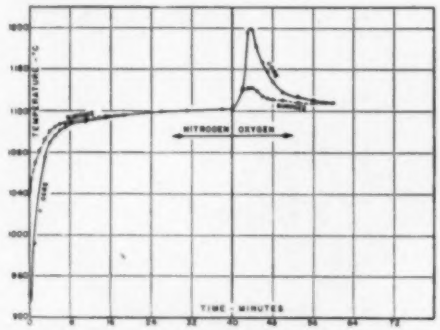


Fig. 9—Time-temperature relationship during heating of a magnetite pellet in nitrogen and subsequent oxidation.

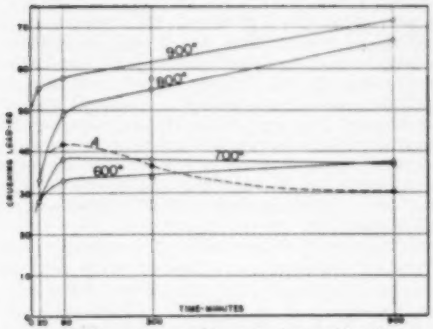


Fig. 10—Strength-time isotherms for ferric oxide pellets heated in air.

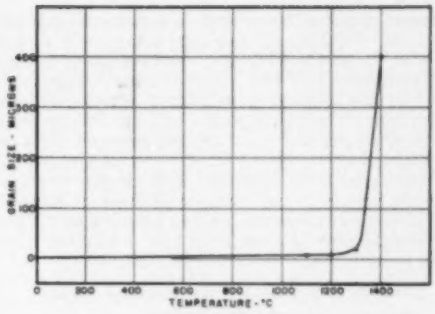


Fig. 11—Grain size-temperature curve for ferric oxide.

diam were heated in air at 700°C for 30 min. After cooling, the pellets were then re-fired in air at various temperatures up to 1200°C, higher temperatures being unattainable with the equipment then available. There was no particular virtue in cooling before re-firing except that the step facilitated laboratory procedure.

Since then the double firing experiments have been performed on identical pellets to a maximum temperature of 1400°C, in a muffle furnace, however, instead of the original tube furnace.

The cycle of heat treatment consisted in heating dried pellets for 2 hr in air at 800°C to obtain at least superficial oxidation of all magnetite particles through to the centers of the pellets. The pellets were then cooled, and subsequently batches were reheated for 30 min at 100° intervals from 1000° to 1400°C. These double-fired pellets were then tested for strength at room temperature. Since the Amsler machine had a limiting compressive load of 300 kg, the higher strength pellets were tested on an hydraulic compression machine.

Microscopic examination of the pellets treated at 800°C showed the oxidation to extend uniformly to the centers. Slime magnetite was completely oxidized, but the larger grains of magnetite were incompletely transformed to hematite. The strength of the pellets was 102.2 kg.

Double firing did not change the strength at 1000°C, but this increased at successively higher temperatures to a maximum, at 1350°C, at which temperature the cold strength was 637.8 kg. At 1400°C the strength decreased to 612.5 kg, see curve B, Fig. 7.

In contrast to these results, identical pellets fired in a single stage to the indicated temperature, i.e., without the pre-firing stage at 800°C, gave a maximum cold strength of only 260 kg at 1400°C, see curve A, Fig. 7.

Pellets fired in pilot-plant furnaces all possess a shell produced by a process analogous to double-firing. As the cold pellet is heated, a wave of oxidation proceeds into the pellet, and initial cementation takes place in the shell. Because of the heat liberated by the strongly exothermic reaction and the decrease in permeability effected by the cementation, after a short period diffusion of oxygen lags behind heat transfer, with the result that the cores are slagged and magnetite is recrystallized before complete oxidation of the pellet occurs. Normally this gives an undesirable duplex structure. Occasionally, however, conditions arise in pilot-plant furnaces which are not well understood, and so-called blue pellets are produced in which there is no megascopic appearance of a duplex structure. However, microscopic examination of polished sections of such pellets always shows that slagging of the siliceous constituents and recrystallization of the magnetite has occurred in the centers, the magnetite eventually having been oxidized to hematite. These blue pellets are rated the best by practical operators.

Thermal Differentials in Pellets During Firing

Much discussion has centered about the magnitude of the temperature differential between core and shell of a pellet during firing. Because pellets occasionally cracked during furnace firing and extruded a more or less liquid mass, it has been assumed that core temperatures are considerably above shell temperatures. Actually, under the oxidizing conditions prevailing in a pilot-plant furnace, the cores possess a lower melting point than the already oxidized shells, and the above argument loses a good deal of its weight.

Ban[5] has shown that during firing the core temperature of magnetite pellets rises above the temperature of the surface. This is true for pellets both with and without included coal, although the temperature differential is greater when coal is present. Fig. 8 shows the measured temperatures of the cores and surfaces of coal-free pellets fired in air and oxygen respectively. For example, the surface temperature of a ¾-in. pellet charged into the 1020°C zone of an electric furnace through which oxygen was passing required 4 min to reach furnace temperature. Both core and surface temperatures then rose to maxima after a total of 6 min in the furnace, when they were respectively 120° and 15° above furnace temperature. Both then dropped to furnace temperature after about 11 min total firing time. A pellet fired in air at somewhat higher furnace temperature than in the last example also exhibited a temperature gradient between core and surface, with a smaller differential, 10°, than with oxygen, 104°. Peak and furnace temperatures were reached after 21 and 40 min, respectively.

Fig. 9 shows the effect of bringing a pellet up to furnace temperature in an inert atmosphere and then admitting oxygen. In nitrogen the core lags behind the surface temperature until equilibrium is reached; then with admission of oxygen both core and surface exceed the furnace temperature, although the gain in both cases is less than when the pellet is

inserted directly into oxygen, and the differential is only 56°. Presumably this last is due to a slower rate of magnetite oxidation, brought about by the recrystallization of that mineral in the nitrogen and also by the lowered permeability effected by slag formation. It seems probable that the temperature differential existing in pilot-plant furnaces is closer to that shown by the air-fired pellet of Fig. 8 than by the pellet fired in oxygen, first because the pellets are brought to temperature more slowly in practice, and second, because partial pressure of oxygen in furnace gases is probably not very different from that employed in this laboratory experiment.

The Effects of Heat Treatment and of Additives on Ferric Oxide

Since the foregoing discussion indicates that the recrystallization and grain growth of hematite is one of the controlling factors in the formation of a strong pellet, an investigation was made to determine the effects of temperature and of selected addition agents on the behavior of this mineral. The initial experiments employed chemical grade ferric oxide, but a closer approach to the practical aspect consisted of an investigation of the combined effects of additives and heat treatment on pellets prepared from the same magnetite concentrates used previously. The selection of the additives was dictated in part by the results obtained with the ferric oxide and in part by a consideration of the requirements of economics and of the reduction process to which the pellets would eventually be subjected.

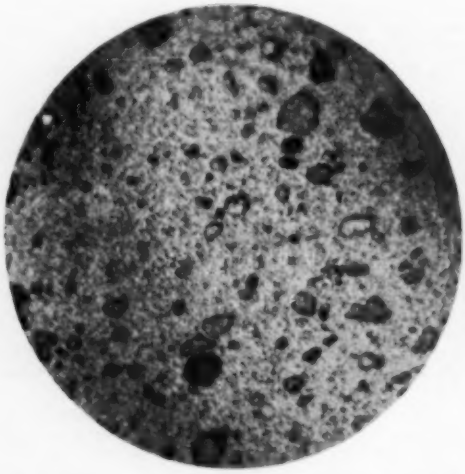
The material used was Baker's analyzed ferric oxide. Particle size was fairly uniform, averaging 0.8 μ. The material contained 99.704 pct Fe_2O_3, 0.170 pct SiO_2, 0.050 pct SO_3, 0.040 pct Mn, and minor amounts, 0.010 or less, of Cu, Zn, NO_3, PO_4, and As.

Reproducible spherical pellets, ¾-in. in diam, were hand-rolled from the dampened oxide, and air-dried prior to firing. Drum pelletizing was difficult to manage, exhaustive of patience and prodigal of time, while briquetting, using either cylindrical or spherical moulds, failed to give unflawed shapes.

The temperatures at which these pellets were fired ranged from 600° up to and including 1200°C, but only those fired below 1000°C could be used for measurement of cold strength, for at 1000° the pellets shrank very considerably and cracked regardless of the heating and cooling rates employed. The effect of holding for various times at temperature was also investigated up to 900°C, with strengths determined after 20, 80, 300, and 900 min firing time. Dry, unfired pellets had strengths ranging from 3 to 4 kg.

The data for this series of experiments are given in Fig. 10, which shows a strength-time isotherm for each temperature investigated. It would have been interesting to obtain grain size measurements for each temperature, but this was impossible below 1100°C because the excessive internal reflection of the small hematite crystals masked the grain boundaries. However, other pellets were heated from 1100° to 1400°C for 30 min, and although it was impossible to measure their strengths because of spalling, measurements of grain size were made and are given in Fig. 11. Figs. 12 to 15 show the striking increase in grain size at these higher temperatures.

When hand-rolled pellets, prepared from a technical grade of ferric oxide, were fired at 1000°C, the strengths were markedly different from those obtained with the purer oxide. They did not stress-crack on cooling, and the shrinkage was much less. As shown by Curve A, Fig. 10, their maximum strength was considerably less than that of the purer pellets fired at 900°C, and at longer firing times the strength actually diminished. Since the particle diameter of both oxides was originally the same, it appeared that the technical grade contained either an additional impurity or more of a specific impurity. Spectrographic analysis showed both oxides to contain the same metallic impurities in about the same quantities with the exception of barium and silicon, which were in larger amounts in the technical grade material. Subsequent chemical analysis gave 3.57 pct Ba, 8.69 pct SiO_2, and 0.069 pct PO_4.



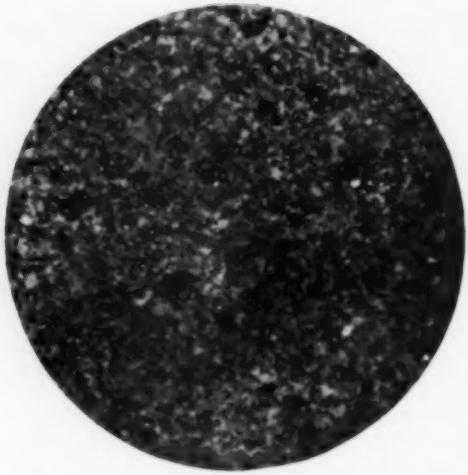


Fig. 12 (Left)—Crossed nicols. Analytical grade ferric oxide compacted and heated for 30 min at 1100°C. Average grain size of hematite, 6 microns. X200. Fig. 13 (Right)—Crossed nicols. The same material shown in Fig. 12, heated for 30 min at 1200°C. Average grain size, 7 microns. X200.

Analytical ferric oxide pellets and similar pellets to which 1 pct $BaCO_3$ was added were fired at 800°C for 300 min. The former had a strength of 53.4 kg, the latter a strength of 8.4 kg. Tests made under identical conditions but with the substitution of 6 pct −200 mesh quartz for the barium carbonate gave a strength of 51.8 kg. These tests indicate that silica, within the limits of the quantity given, has little effect on the pellet strength, but that barium has an extraordinary effect in lowering strength.

Although the writers have not investigated this matter further, it appears that relatively small amounts of certain additives, or even impurities occurring naturally in the ore, might exert a deleterious effect on pellet strength.

A number of inorganic additives were considered which might improve strength of pellets. Those finally selected were calcium oxide, magnesium oxide, sodium fluoride, and sodium carbonate. It was hoped that these additives would improve the physical properties of the pellets, either by accelerating grain growth or by reducing the temperature at which optimum strength could be obtained.

Hand-rolled pellets were prepared from analytical grade oxide containing 1 and 5 pct of the selected additives, and were fired in air for 30 min in an electric muffle furnace at temperatures ranging from 800° to 1400°C at 100° intervals.

At 800°C pellets containing borax cracked and spalled, and those with sodium fluoride explosively disintegrated. At 900°C the only pellets remaining intact were those containing lime and magnesium oxide. At 1000°C the lime and magnesia-containing pellets showed evidence of surface cracking, and at higher temperatures all pellets showed the fissures which occurred in ferric oxide pellets fired to corresponding temperatures without an additive. Since this was normal under the circumstances, the tests were considered capable of furnishing useful information, and it was decided to investigate further the effect of adding lime and magnesia to eastern Mesabi magnetite pellets.

The Effect of Firing on Magnetite Pellets Containing Added Lime and Magnesia

Six sets of drum-rolled magnetite pellets were prepared. The first two sets contained 5 pct calcium oxide added during rolling. One set was used for direct firing, and the other for re-firing at elevated temperatures following an oxidizing heating at 800°C for 2 hr. The third set contained 10 pct added lime and was direct-fired.

The fourth and fifth contained 5 pct magnesium oxide added during rolling, and the sixth contained 10 pct added magnesia. Originally the intention was to fire the magnesia-containing pellets according to the same procedure outlined above for the lime-bearing pellets, but it was found that they required careful heating up to 500°C to prevent them from explosively disintegrating upon being heated to 800°C or above. The procedure was accordingly modified to meet these requirements. This initial treatment undoubtedly permitted partial oxidation and therefore tended to minimize the differences between single-fired and double-fired pellets.

Figs. 16 and 17 show the strengths of pellets containing lime and magnesia respectively and fired according to the procedure given above. They should be compared with Fig. 7.

Single-fired pellets containing lime showed no significant difference in strength compared with straight magnetite pellets fired in the same way. The maximum strength was obtained with pellets pre-oxidized at 800°C and refired at 1300°C, but it was only about 2/3 the maximum strength shown by Curve B of Fig. 7. Abrasion resistances of pellets were compared qualitatively by means of rubbing pellets together, an insufficient number being available for a tumbling test. On the basis of this crude test the abrasion resistance of those pellets containing 5 pct lime, pre-oxidized at 800° and fired at 1300°C, was markedly superior to that of pellets without lime, pre-oxidized at 800° and fired at 1200°, 1300°, and 1400°C.

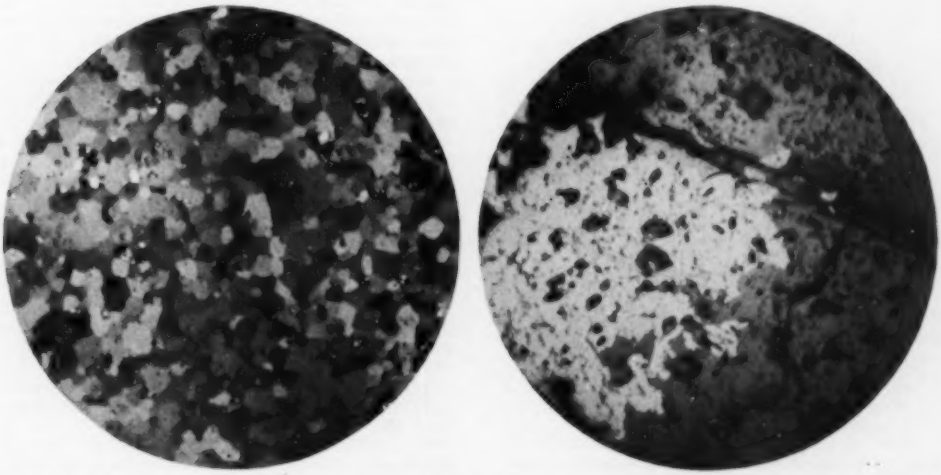


Fig. 14 (Left)—Crossed nicols. The same material shown in Fig. 12, heated for 30 min at 1300°C. Average grain size, 20 microns. X200. Fig. 15 (Right)—Crossed nicols. The same material shown in Fig. 12, heated for 30 min at 1400°C. Average grain size, 400 microns. X200.

Calcium oxide does not react with magnetite, but it unites with ferric oxide to form calcium ferrites. According to the equilibrium diagram[2] for the system CaO-Fe_2O_3, for 5 pct and 10 pct by weight of CaO, calcium ferrite, CaO Fe_2O_3, is in equilibrium with hematite below 1200°C. The presence of silica complicates the situation because this reacts at high temperatures with both lime and ferrite to form silicates. Above 1250°C hematite forms magnetite.

Microscopic examination of the fired pellets containing lime showed that oxidation of the magnetite occurred more readily at low temperatures than in the absence of lime, presumably because the additive preserved the original permeability. Ferrite formation was rapid at 1000° and complete at 1100°C, but at 1200°C and higher the silicates reacted with ferrite to form slag and hematite.

Magnesium oxide does not react with magnetite, but it unites with ferric oxide to form a spinel which is isomorphous with magnetite. As earlier work indicated that magnetite exhibited grain growth commencing at about 800°C and becoming pronounced at 1000°C, it was thought possible that the spinel structure could be maintained in an oxidizing atmosphere by the introduction of magnesium oxide, the assumption being that the magnesium spinel would exhibit grain-growth properties similar to those of magnetite.

Experimentally, however, it was found that the temperature required to replace hematite by magnesium spinel was so high, 1000°C or more, and the rate of replacement so slow that nothing was gained by using this additive. As shown by Fig. 17, the maximum strengths were not high, and simple tests showed that the abrasion resistance of these pellets was inferior to those of both the lime-magnetite and the magnetite pellets.

Summary

Under laboratory conditions, maximum cold strength was obtained from magnetite pellets first by a superficial oxidation of all included magnetite grains at moderate temperatures and then by an increase of the firing temperature to promote grain growth of the resulting hematite. Such treatment affects uniformity of grain size and homogeneity of structure, the contraction crack between core and shell, so characteristic of pellets which have been brought rapidly to temperature in an oxidizing atmosphere, being subordinated or entirely eliminated. Excessive firing temperature was deleterious to strength because of thermal decomposition of the hematite, accompanied by slagging, breakdown of the continuous oxide network, and creation of blowholes due to the liberation of oxygen.

Neither calcium nor magnesium oxides added during rolling gave fired strengths comparable to those obtained from the unadulterated magnetite concentrate pellets, although calcium oxide conferred a measure of abrasion resistance.

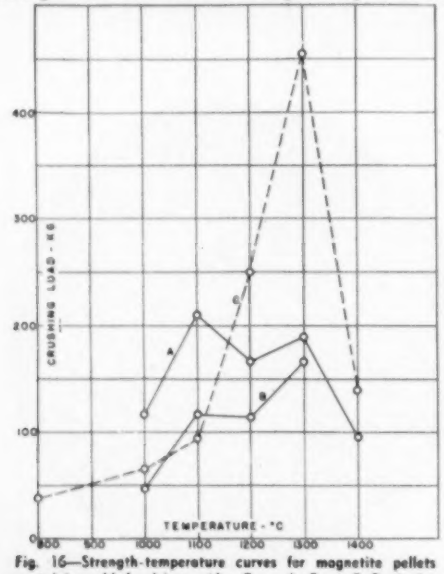


Fig. 16—Strength-temperature curves for magnetite pellets containing added calcium oxide. Curve A: 5 pct CaO, single-fired. Curve B: 10 pct CaO, single-fired. Curve C: 5 pct CaO, oxidized in air for 2 hr at 800°C and refired at the temperatures indicated.

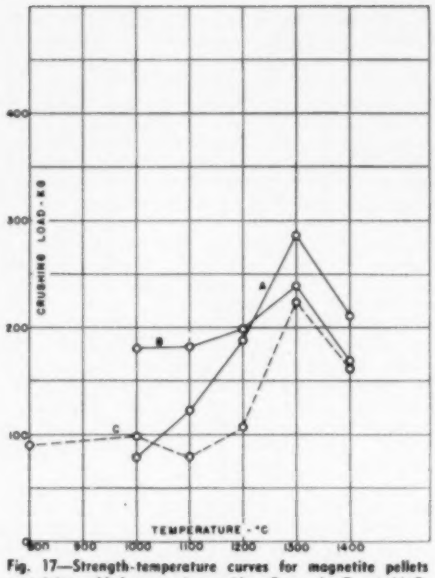


Fig. 17—Strength-temperature curves for magnetite pellets containing added magnesium oxide. Curve A: 5 pct MgO, preheated in air at 500°C and reheated to the temperatures indicated. Curve B: 10 pct MgO, preheated in air at 500°C and reheated to the temperatures indicated. Curve C: 5 pct MgO, preheated in air for 2 hr at 800°C and refired at the temperatures indicated.

Acknowledgments

The authors' thanks are extended to Dean Thomas L. Joseph for his continued interest and help, to Dr. Wladimir Philippoff for many suggestions made during the progress of this work, and to the Allis-Chalmers Manufacturing Co. for donating equipment and for sponsoring a fellowship for the junior author through which a major part of this work was accomplished.

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Effects of Alkalinity on the Flotation of Lead Minerals

by Marston G. Fleming

General theories of alkali depression are disproved by this paper, which shows that its operation depends upon the nature of the mineral-collector system involved. Three mechanisms of alkali depression are examined and limitations of critical pH as a fundamental factor in flotation are discussed.

CRITICAL pH has been defined by Wark[1] as that pH value below which a mineral will float and above which it will not float in solutions containing a given concentration of collector but free from other depressants or activators. The relationship between mineral, collector concentration, and pH is expressed in the form of a critical pH curve determined by the captive bubble method and is generally regarded as being fundamental to the system involved. The mechanism of alkali depression is currently in dispute, two general theories having received authoritative support. The view expressed in Taggart's chemical theory of flotation[2] is that in the presence of a critical concentration of hydroxyl ions, all the metal ions available at the surface of the mineral are present as anions of the form $[M(OH)_x]^{-y}$. The formation of the necessary metal xanthate is thus precluded and, furthermore, any metal xanthate coating previously formed, at a lower pH, will be dissolved. Gaudin[3, 4] and the Melbourne school under Wark[5] hold that alkali depression is caused by the preferential adsorption of hydroxyl ions by the mineral surfaces.

This paper describes an investigation of the mechanism of alkali depression in each of the three systems: cerussite-aqueous carbonate-xanthate, cerussite-xanthate, and galena-xanthate.

Alkali Depression of Cerussite by Control of Carbonate Ion Concentration

Wark and Cox[6] have shown that in the presence of Na_2CO_3, a higher concentration of xanthate is required to effect bubble contact with a cerussite surface than in carbonate-free solutions. To examine this depression of cerussite by sodium carbonate, a series of captive bubble tests was carried out on cerussite specimens in solutions containing 100 mg per liter of potassium ethyl xanthate and varying concentrations of sodium carbonate. The pH was adjusted by the addition of dilute hydrochloric acid. For purposes of comparison a standard critical pH curve was determined for cerussite with potassium ethyl xanthate in solutions containing no added carbonate. In the latter case pH adjustment was made with sodium hydroxide. The experimental method employed in both series was broadly similar to that described by Wark and Cox;[7-9] a number of refinements suggested by Sutherland[10] were adopted, and the procedure was further developed and modified to suit the particular requirements of the investigation.

The relationship between pH and the concentration of potassium ethyl xanthate necessary to induce bubble contact at cerussite surfaces is shown in Fig. 1. The relationship between pH and the concentration of sodium carbonate necessary to prevent bubble contact at cerussite surfaces in the presence of 100 mg per liter of potassium ethyl xanthate is shown in Fig. 2. Comparison of the two curves shows that at a collector concentration of 100 mg

Table I. Composition of Solutions Containing Na_2CO_3 Just Sufficient to Prevent Contact at Cerussite Surfaces in the Presence of 100 Mg Per Liter. Potassium Ethyl Xanthate

pH	Concentration in Mols Per Liter			
	Na^+	Cl^-	HCO_3^-	$CO_3^=$
8.1	0.0263	0.0135	0.0146	0.71×10^{-4}
8.2	0.0255	0.0136	0.0126	0.78×10^{-4}
8.3	0.0232	0.0114	0.0116	0.89×10^{-4}
8.4	0.0200	0.0095	0.0103	1.00×10^{-4}
8.5	0.0168	0.0075	0.0089	1.08×10^{-4}
8.6	0.0134	0.0060	0.0072	1.10×10^{-4}
8.7	0.0110	0.0052	0.0056	1.08×10^{-4}
8.8	0.0088	0.0043	0.0044	1.04×10^{-4}
8.9	0.0072	0.0037	0.0035	1.07×10^{-4}
9.0	0.0060	0.0032	0.0026	1.00×10^{-4}
9.1	0.0050	0.0027	0.0021	1.03×10^{-4}
9.2	0.0038	0.0023	0.0014	0.87×10^{-4}
9.3	0.0029	0.0017	0.0010	0.80×10^{-4}
9.4	0.0023	0.0013	0.0008	0.81×10^{-4}
9.5	0.0015	0.0009	0.0005	0.59×10^{-4}
9.6	0.0008	0.0003	0.0004	0.59×10^{-4}

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per liter KEX, the presence of 0.02 molar Na_2CO_3 lowers the critical pH from 9.7 to 8.0.

The carbonate solutions contained Na^+, K^+, Pb^{++}, H^+, OH^-, Cl^-, $CO_3^=$, HCO_3^- and EX^- and, at the concentrations used, dissociation may be assumed to be complete. Na, K, H, Cl, and EX were measured quantities. By introducing these known values into the statement of electrical neutrality for the system and combining this with the second dissociation constant of carbonic acid and the ionic product of water, it is possible, with a little algebraic manipulation, to calculate the missing factors. In Table I are listed the concentrations of the various ions present in the solutions which correspond to selected points along the curve in Fig. 2. The quantities tabulated are, therefore, the concentrations present when just sufficient sodium carbonate has been added to prevent the contact of an air bubble at the surface of cerussite in the presence of 100 mg per liter of potassium ethyl xanthate. With the exception of the carbonate ion itself, the concentrations of the ions vary widely along the length of the curve. The concentration of $CO_3^=$ is, however, remarkably constant and the curve, in effect, is a line of constant carbonate ion concentration, solutions to the right of the curve having a greater concentration and solutions to the left a smaller concentration of this ion than the constant. Variations at high pH may be explained by the fact that these solutions are approaching the point where alkalinity itself is the critical agent. The curve becomes almost asymptotal at pH 8 and variations at low pH are probably due to magnification of small errors by the steep slope of the curve.

The critical carbonate ion concentration for air-cerussite contact in the presence of 100 mg per liter of potassium ethyl xanthate may, then, be taken as 1.0×10^{-4} mols per liter at 25°C.

Interpretation of Results: Latimer[11] gives the following value for the solubility product of lead carbonate at 25°C:

$$(Pb^{++})(Co_3^=) = 1.5\times10^{-13} \qquad [1]$$

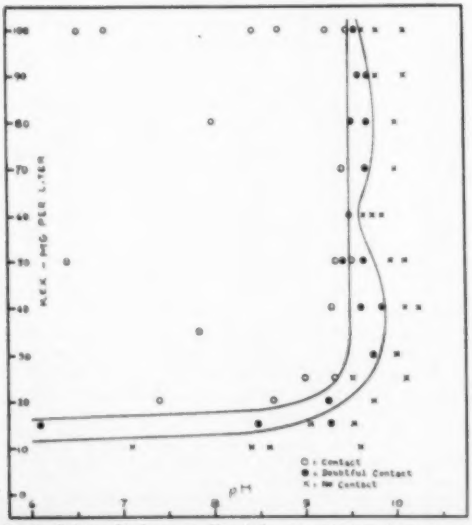

Fig. 1—Relationship between pH and the concentration of potassium ethyl xanthate necessary to induce contact at cerussite surfaces.

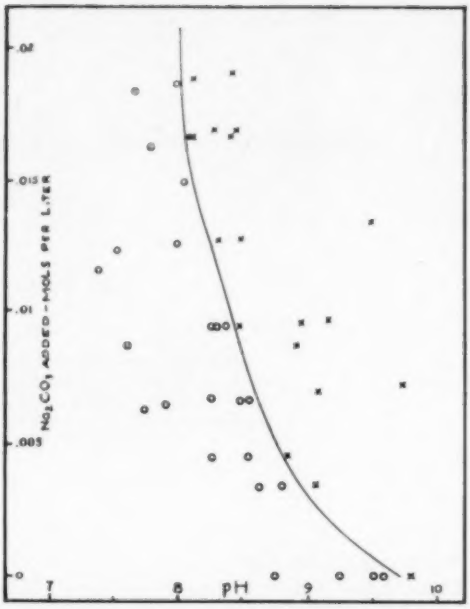

Fig. 2—Concentration of sodium carbonate necessary to prevent contact at cerussite surfaces.

In the presence of the critical carbonate ion concentration determined above the solubility of the lead ion becomes

$$(Pb^{++}) = \frac{1.5\times10^{-13}}{1.0\times10^{-4}} = 1.5\times10^{-9} \text{ mols per liter}$$

Taggart and Hassialis[12] have determined the solubility product of lead ethyl xanthate to be 6.7×10^{-16}

$$(Pb^{++})(EX^-)^2 = 6.7\times10^{-16} \qquad [2]$$

Potassium ethyl xanthate to the amount of 100 mg per liter corresponds to a xanthate ion concentration of 6.25×10^{-4}; by a substitution of this value in the solubility product, the saturation value for lead ion in the xanthate solutions tested may be obtained.

$$(Pb^{++}) = \frac{6.7\times10^{-16}}{(6.25\times10^{-4})^2} = 1.7\times10^{-9}$$

This figure is in close agreement with the lead ion concentration present in aqueous carbonate solutions that contain the critical carbonate ion concentration. Therefore, there is strong justification for stating that in the system xanthate-aqueous carbonate-cerussite, the carbonate ion is the controlling factor in determining the critical pH above which an air bubble cannot make contact with the mineral surface and the function of the carbonate is to reduce the solubility of the cerussite to a point at which the concentration of lead ions is equal to or less than the saturation value for lead xanthate in the solution.

Alkali Depression of Cerussite by Mineral Decomposition

To study the reaction between cerussite and alkali a galvanic cell capable of registering small

changes of lead ion concentration was employed:

$$Pb \mid Pb^{++}\ (aq) : KCl\ (aq\ saturated), HgCl_2 \mid Hg.$$

The lead electrode was specially purified rod, 99.9988 pct Pb, supplied by Johnson Matthey & Co. The test solution vessel contained a mechanical stirrer and a glass electrode which, by a special switching arrangement, could replace the lead electrode in the circuit so that EMF and pH could be measured alternately. The complete cell was suspended in a thermostatic bath at 25°C. The lead ion concentration was measured in aqueous solutions in contact with solid, freshly ground cerussite and the alkalinity of the solution was progressively increased by additions of potassium hydroxide. After each addition of alkali the mixture was left for 55 min with the glass stirrer producing a gentle agitation. At the end of this time the pH and EMF were recorded and the next addition of alkali was made.

The results are shown in Fig. 3 where lead ion concentration is plotted against hydrogen ion concentration on a negative logarithmic basis. There is very little change in lead ion concentration until the pH approaches 9 when there is a sharp inflection; thereafter, the lead ion concentration decreases rapidly as the alkalinity is increased. The results do not represent true equilibrium conditions for either hydrogen or lead ion concentrations, as these could only have been obtained with intervals of 2 to 3 weeks between KOH additions. However, the 1-hr interval allowed for the establishment of partial equilibrium was, as the work showed, sufficient to smooth out the initial instability brought about by the addition of alkali and was actually more representative of the conditions obtaining in flotation test work than would have been the case had full, stable equilibrium been established.

Interpretation of Results: It has already been shown that, in cerussite flotation, the lead ion concentration is a controlling factor in determining the critical pH. The sharp inflection point which occurs near pH 9 in the curve shown in Fig. 3 is, therefore, undoubtedly significant. It is suggested that this point marks the pH at which, under the conditions

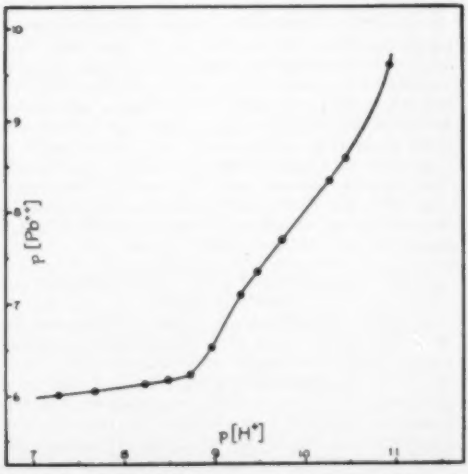

Fig. 3—Cerussite. Relationship between lead ion concentration and hydrogen ion concentration.

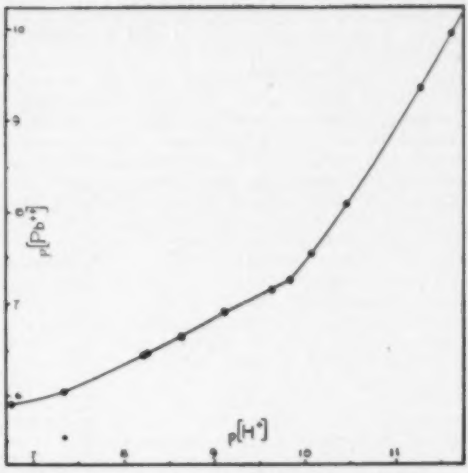

Fig. 4—Lead ethyl xanthate. Relationship between lead ion concentration and hydrogen ion concentration.

of the test, the formation of basic lead carbonate begins. Thus:

$$3Pb\,CO_3 + 2\,OH^- = Pb_3(CO_3)_2(OH)_2 + CO_3^= \quad [3]$$

In alkaline solutions, basic lead carbonate does not dissociate to yield free lead ions but, as Randall and Spencer have shown,[12] reacts with more hydroxide in the following manner:

$$Pb_3(CO_3)_2(OH)_2(solid) + 7\,OH^- = 3HPbO_2^- + 2CO_3^= + 3H_2O \quad [4]$$

The presence of excess $CO_3^=$ over and above that produced by the dissociation would repress these reactions and, indeed, in carbonate solutions at low alkalinity, basic lead carbonate is dissociated with the formation of Pb^{++} and HCO_3^- [13]:

$$Pb_3(CO_3)_2(OH)_2(solid) + 4H_2CO_3(aq) = 3Pb^{++} + 6HCO_3^- + 2H_2O \quad [5]$$

Carbonate ion thus acts in two ways to control the concentration of lead ion and the formation of the desired hydrophobic coating of lead xanthate on the surface of cerussite. 1—As has been shown, it has direct influence on the dissociation:

$$PbCO_3(S) = Pb^{++} + CO_3^= \quad [6]$$

2—It represses the formation of basic lead carbonate. In solutions to which no carbonate salt has been added the critical pH for cerussite flotation would correspond to the conversion of a limiting proportion of the mineral surface to basic lead carbonate. In solutions containing added carbonate ion, critical pH might represent the formation of basic lead carbonate as described above or it might represent the point at which the dissociation of cerussite has been reduced, through the mechanism of eq 6, until the concentration of lead ions is equal to or less than saturation value for lead xanthate in the solution.

Alkali Decomposition of Lead Xanthate

Using the galvanic cell and experimental procedure previously described, the influence of increasing concentrations of potassium hydroxide

Table II. Captive Bubble Tests with Galena

Test	Preconditioning: KEX Mg per Liter	Preconditioning: pH	Preconditioning: Time, Min	Rinse Prior to Transfer	Test Solution: KEX Mg Liter	Test Solution: pH	Min After Pre-Conditioning	Captive Bubble Reaction
1	25	7.15	30	Water	0 —	6.7	0 — 120	Strong contact continuing
2	25	7.15	30	Water	0 —	6.7	0 — 2	Strong contact
				None	0 —	12.2	2 — 3	Sticking
				Not transferred	0 —	12.2	3 — 4	No contact
				Water	0 —	6.7	4 — 10	No contact
				None	25 —	6.7	10 — 12	Strong contact in ½ min
				Water	0 —	12.2	12 — 14	Sticking
				Not transferred	0 —	12.2	14 — 15	No contact
				Water	0 —	6.7	15 — 30	No contact
3	25	7.15	30	Water	0 —	7.0	0 — 2	Strong contact
				None	0 —	11.9	2 — 4	Sticking
				Water	0 —	7.0	4 — 6	Sticking
				None	0 —	11.9	6 — 8	Sticking
				Not transferred	0 —	11.9	8 — 34	No contact
				Water	0 —	7.0	34 — 150	No contact
4	100	7.2	30	Water	0 —	6.5	0 — 2	Strong contact
				None	0 —	11.7	2 — 9	Contact
				Not transferred	0 —	11.7	9 — 10	Sticking
				Water	0 —	6.5	10 — 11	Sticking
				None	0 —	11.7	11 — 12	Sticking
				Not transferred	0 —	11.7	12 — 13	No contact
				Water	0 —	6.5	13 — 30	No contact
5	100	7.2	30	Water	0 —	6.8	0 — 2	Strong contact
				None	0 —	10.7	2 — 25	Contact
				Not transferred	0 —	10.7	25 — 62	Sticking
				Not transferred	0 —	10.7	62 — 64	No contact
				Water	0 —	6.8	64 — 100	No contact
6	25	7.26	30	Not transferred	25 —	7.26	0 — 2	Strong contact
				None	25 —	11.8	2 — 4	Contact
				Not transferred	25 —	11.8	4 — 21	Sticking
				Not transferred	25 —	11.8	21 — 31	No contact
				Water	0 —	6.9	31 — 41	No contact
				None	25 —	7.26	41 —	Strong contact after 1 min
7	25	11.83	30	Not transferred	25 —	11.83	0 — 2	No contact
				Water	0 —	6.7	2 — 22	No contact
				None	25 —	7.1	22 —	Strong contact after 1 min

upon the lead ion concentration of aqueous solutions in contact with solid lead ethyl xanthate was measured. The results are shown in Fig. 4 where p (Pb^{++}) is plotted against p(H^+). The curve indicates a gradual decrease in lead ion concentration up to pH 10 which marks an inflection and, thereafter, a sharp reduction of (Pb^{++}) as the alkalinity is further increased.

During these experiments it was observed that upon the addition of KOH which raised the pH from 9.82 to 10.07, the solution, which had hitherto been colorless, assumed a distinctly yellow shade and the solid lead xanthate became tinged with brown. Further additions of alkali accentuated this change and, at the end of the test, the liquid was distinctly yellow and the solid residue had turned dark brown. This was confirmed by a series of tests in which solid lead ethyl xanthate was placed in solutions of varying alkalinity and changes were observed over a period of 20 hr. These left no doubt that the decomposition of lead ethyl xanthate is a gradual process the rate of which depends upon the concentration of alkali in the solution. Some of the solid, dark brown reaction product was prepared by a method which removed all soluble matter but avoided the loss of any solids. Analysis showed this to be pure lead sulphide.

Interpretation of Results: These results show that under conditions similar to those in which flotation testing is carried out, lead ethyl xanthate is decomposed by alkali with the formation of lead sulphide as a reaction product. The curve in Fig. 4 is, in effect, a record of this decomposition. Below pH 10 the reaction is slow and, under the usual conditions of flotation research, probably not critically important. Above pH 10 the reaction becomes more rapid and apparently proceeds to completion. Although complete conclusions cannot be established from the available data, the following three-stage reaction is suggested:

$$(C_2H_5-O-C(=S)-S)_2\ Pb + 4KOH \rightarrow 2C_2H_5-O-C(=S)-S-K + K_2PbO_2 + 2H_2O \quad [7]$$

$$C_2H_5-O-C(=S)-S-K + 2KOH \rightarrow C_2H_5-O-C(=O)-S-K + K_2S + H_2O \quad [8]$$

$$K_2PbO_2 + K_2S + 2H_2O \rightarrow \underline{PbS} + 4KOH \quad [9]$$

In the first stage the lead ethyl xanthate is decomposed by alkali into soluble plumbate, alkali xanthate, and water. An alteration of alkali xanthate into monothiocarbonate, potassium sulphide, and water is known to occur in alkaline solutions, but this reaction would not be favored were it not for removal of potassium sulphide from solution by interaction with the plumbate and precipitation of lead sulphide as shown in eq 9. Under the circumstances, however, reactions eq 8 and eq 9 can proceed until the plumbate is used up, and simultaneous production of hydroxide from eq 9 will help to maintain the alkalinity of the system.

Alkali Decomposition of Collector Coatings at Galena Surfaces

A series of captive bubble tests were carried out in which galena was first conditioned in neutral potassium ethyl xanthate solutions and then transferred to solutions of varying pH. In transferring from one solution to another the specimen was held face down in glass tongs while 50 ml of solution was poured over it as a rinse. While they were immersed in the test solutions, the mineral specimens were tested for bubble contact at very short intervals, usually every 30 to 60 sec. Results are given in

Table II. The time required for rinsing and transferring the specimens was probably never longer than 10 sec and is omitted from the timetable which refers only to the period in which the specimens were immersed in the test solution. Temperature was maintained at 25°C.

Interpretation of Results: The results of these captive bubble tests may be summarized as follows:

1—The collector coating formed by potassium ethyl xanthate on galena surfaces is stable in neutral solutions, Test 1. This fact is well known to flotation investigators.

2—A preformed xanthate coating on galena loses its hydrophobic character in alkaline solutions, Tests 2 to 5.

3—The alteration of such a preformed coating by alkaline solutions is gradual, the rate being proportional to the alkalinity of the solution, Tests 2 to 5. In strongly alkaline solutions, the mineral surface becomes hydrophilic very quickly, Test 2. In solutions whose pH is close to the critical pH for galena, the change to a wettable surface may take as much as 1 hr, Test 5.

4—A collector coating which is formed by preconditioning in 100 mg per liter KEX solution appears to be more resistant to alkaline solution than one formed in 25 mg per liter KEX solution, Tests 3 and 5.

5—Failure to effect bubble contact with a precoated galena surface in alkaline solutions is due to a definite alteration of the coated surface and not simply to the concentration of hydroxyl ions at the solid-liquid interphase. At every stage in the alteration of the surface the same degree of wettability is found when the mineral is removed from the alkaline solution, washed in running distilled water, and tested in water such as existed in the alkaline solution immediately before its transfer, Tests 2 to 7.

6—The presence of alkali xanthate in the solution retards but does not prevent the alteration of a preformed xanthate coating by alkaline solutions, Tests 3 and 6.

7—The action of alkaline solutions on a galena surface, whether precoated with xanthate or not, does not prevent its ready acceptance of a new xanthate coating when it is transferred to a neutral xanthate solution, Tests 1, 6, and 7.

These observations suggest that the collector coated surface of galena is, in effect, lead xanthate and that the depressant action of high pH in flotation is due to the decomposition of that surface as described in this paper under Alkali Decomposition of Lead Xanthate. The relation of items 1 to 5, above, to this hypothesis is self-evident. Item 6 is explained by reference to eq 7, from which it can be seen that the decomposition of lead ethyl xanthate by alkali will be repressed by the presence of alkali xanthate in the solution. Item 7 derives clearly from the fact that the decomposition results in the deposition of fresh lead sulphide upon the surface of the mineral, which is thus well conditioned for collector-coating in a neutral or slightly alkaline solution.

Summary and Conclusions

The tests described show that no general theory of alkali depression is tenable, for the mechanism of depression depends upon the nature of the mineral-collector system concerned. It has been shown that the action of carbonate-free alkaline solutions in preventing the flotation of cerussite may be independent of the collector and may be due entirely to decomposition of the mineral surface. This type of alkali depression is not necessarily confined to cerussite but may operate in any case where the basic product does not yield simple metal ions. When the xanthate collection of cerussite is carried out in solutions containing an appreciable amount of soluble carbonate, the alkali has been shown to exert its depressant effect indirectly through control of the carbonate ion concentration. In this case there is no true alkali depression, for the formation of a collector coating is prevented by solubility-product control of a metathetical reaction. A third type of alkali depression which has been studied in this research operates through the decomposition of the collector coating by alkaline solutions. This is almost certainly a critical influence in the xanthate flotation of galena and, possibly, in the flotation of other metal sulphides with anionic collectors. It would be extremely dangerous, however, to draw general conclusions from results based entirely on experiments with lead xanthate and galena, for the behavior of the other principal sulphide minerals, those of copper, iron and zinc, in xanthate flotation is considerably more complicated than that of galena. With other mineral-collector systems alkali depression may operate through an entirely different mechanism from any described above.

When alkali depression is caused by decomposition of collector coatings it is apparent from an examination of the curve in Fig. 4 that the term *critical pH* can be relative only. The slope of this curve together with the results given in Table II show that decomposition of the metal xanthate is a continuous process which, although accelerated above a certain pH, proceeds slowly even in weakly alkaline solutions. Time, as well as hydroxyl ion concentration, is thus an essential factor, and the correlation of different critical pH curves is dependent, to some degree at least, upon the conditioning period. The close standardization of conditions for captive bubble testing by investigators at Columbia and Melbourne Universities and the almost universal adoption of their methods by other workers has tended to hide this effect.

The standard method in such tests employs a conditioning period of 30 min. Where the agency of alkali depression is decomposition of the metal xanthate, it would be expected that under standard conditions the critical pH curve corresponds to the locus of the points at which accelerated decomposition begins. With conditioning periods shorter than standard the position of the curve would probably remain much the same. With longer conditioning periods, however, it would be expected that critical pH would be lower and that the resultant curves would probably lie inside, or slightly to the left of the standard curve.

Similarly, it would be reasonable to expect that the rate of decomposition would be influenced by temperature in such a way that for a given xanthate concentration the critical pH would vary inversely as the temperature of the test. Wark[10] has shown this to be the case with critical pH values for galena, pyrite, and chalcopyrite in 25 mg per liter potassium ethyl xanthate solutions. For galena, the critical pH value is given as 10.8 at 10°C and 9.7 at 35°C.

The time required for the destruction by alkaline solutions of the metal-xanthate coating on mineral surfaces would depend upon the thickness of that coating. With cerussite and anglesite, both rela-

tively soluble minerals, there is strong evidence, supported by the results given above under Alkali Depression of Cerussite by Control of Carbonate Ion Concentration, that the collector coating is produced metathetically and may have an appreciable thickness. Similarly, it might be expected that the oxidized surfaces of sulphide minerals would react with xanthates to form a heavier layer of metal xanthate than would be the case with fresh, unoxidized surfaces. If the equilibrium were not complicated by other factors, such as decomposition of the oxidized mineral surface itself by alkali as in the case of lead carbonate, it is reasonable to suppose that a mineral surface capable of forming a heavy coating of metal xanthate would be less susceptible to alkali depression than one which would normally form a monomolecular film. In this connection it is interesting that Sutherland and Plante[16, 17] found that oxidizing the surface of a sulphide mineral changed its response to ethyl xanthate and led to greater tolerance to the depressant action of cyanide and alkalis.

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Theory, Scale-Up, and Operating Variables of the Peterson Top Feed Reservoir

by R. J. Piros, R. A. Brusenback, and D. A. Dahlstrom

DEWATERING fine coal has been a serious problem to many operators who desired to wet-wash finer sizes and maintain high recovery. Centrifugal driers have become popular for this purpose for sizes between $\frac{3}{8}$ in. and 14 mesh. Below 14 mesh the perforated screen centrifuges lose a large percent in the effluent solid bowl types that do retain −14 mesh coal, produce excessive degradation, tend to retain the highest ash fraction in any size fraction, and are subject to abrasion, which results in higher maintenance.

In 1949 Truax-Traer Coal Co. planned a cyclone installation at their Ceredo preparation plant to recover coal down to 150 mesh with approximately a 14-mesh top size. Since available centrifugal equipment was not considered satisfactory, an investigation of rotary vacuum filters was begun in cooperation with the Peterson Filters and Engineering Co. to determine the suitability of the Peterson Top Feed Reservoir (TFR) filter. This investigation, which is described below, has resulted in a highly satisfactory installation of two 6x10-in. Peterson TFR filters that have been in operation over a year.

After some preliminary laboratory batch vacuum filter tests, a pilot plant double-drum Peterson TFR filter was installed at Ceredo to prove the feasibility of vacuum filtration and determine optimum conditions. The full scale units were then installed on the basis of the pilot plant results. This was followed by an extensive research program on vacuum filtration in the laboratory at Northwestern University to investigate basic design and operating variables and their influence on final moisture content of the filter cake. A method of scale-up from laboratory batch tests to the full-size unit was also proposed and proved. Because of the importance of dewatering fine coal today, it was felt that the results of the three steps should be made available for the use of others facing this problem. The work at Northwestern is given in a theoretical section following the discussion of the pilot plant and full-scale operation.

Fig. 1 is a flow diagram of the cyclone and filter installation. The feed is drawn from below the

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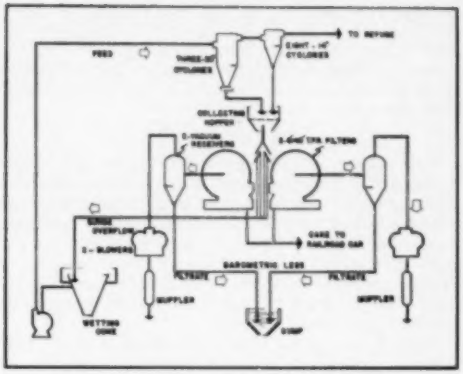

Fig. 1—Flow diagram of cyclone and filter installation.

surface of two raw coal wetting cones and pumped to a battery of three 20-in. and eight 14-in. cyclones arranged so that the overflow of the 20-in. forms the feed for the 14-in. cyclones. No intermediate pump is used. The underflow from the eleven cyclones discharges into a collecting hopper which splits the feed to the two filters or applies it to either one when desired. The solids content of the filter feed is regulated by a back-pressure valve on the overflow from the 14-in. cyclones and by adjustable underflow nozzles on the 20-in. units. The thickened slurry drops into a top feed reservoir formed by the filter drum and a slightly off-vertical steel sheet. A seal is made at the horizontal centerline of the drum by a heavy rubber pad and a bed of coal. The filter cake is formed in this reservoir by the application of vacuum and is carried on the filter screen as the drum revolves toward the scraper. The filter valve bridge is adjusted so that the cake is discharged under vacuum. Moisture is removed by the pressure differential across the cake and by the intense vibration generated by electro-magnetic vibrators mounted on the inside of the drum. The drum is set on rubber pads and is composed of an inner stainless steel drum deck with a stainless steel B-Z tri-rod screen with ¼-mm slot. The space between the drum deck and the screen is divided into narrow channels through which the filtrate and air are withdrawn. These two phases are separated in a 4-in. diam vacuum receiver with a barometric leg for filtrate removal. Vacuum is created by two 18x 27 RCD Roots-Connersville positive displacement blowers.

Operating Results

Initial operation of the filter installation was begun in December 1950. Since that time only one minor part, the Micarta valve bridge, has been replaced, owing to abrasive or structural failure. Several minor changes have been made to improve the operation, e.g., new scrapers have been designed and tension springs for the scrapers have been installed. These improvements are now standard equipment on all models. The only maintenance work that has been done is to tighten the stainless steel hold-down bolts on the screen to the deck to prevent air leakage. This was done after about three months of operation. The screen itself shows little sign of wear, and the scrapers, which are the points of greatest abrasion, are in good condition. No trouble is expected or has occurred at any other point. The relatively slow movement of all parts has overcome the high maintenance problem of most mechanical dewatering devices.

The fine coal circuit is subject to wide fluctuations in feed rate and size distribution because of the variety of coal processed by the plant. For this reason, it is necessary to employ an operator for the filter and cyclones, although he has been able to perform considerable clean-up and general plant maintenance when feed conditions are uniform. Because of these wide fluctuations, the units have handled over 100 tons per hr for short periods. However, best moisture results are obtained at solid rates under 80 tons per hr. Table I gives operating data for several tests under varying conditions.

The average surface moisture for 23 tests on West Virginia No. 2 gas was 14.73 pct at an average capacity of 59.2 tons per hr for the two 6x10's. The average size-consists of the filter cake for these 23 tests is given in Table II. Samples of the filtrate exhibit a very low solids content and contain pri-

Table I. Operating Data for Two 6x10-Ft Peterson TFR Filters, West Virginia No. 2 Gas Coal, Truax-Traer Coal Co., Ceredo, West Va.

Date	Surface Moisture, Pct	Dry Ash, Pct	Vacuum, In. Mercury	Drum Speed, Min Per Rev	Cake Thickness, In.	Air Rate Standard, Cfm	Calculated Solids, Tons Per Hr*	—200 In. Cake, Pct	Vibration Intensity, Pct
12-15-50	15.61	7.98	8	2.0	2⅞	4800	39.4**	5.6	75
12-19-50	15.33	8.02	11	2.0	3	4040	53.0	4.9	75
12-21-50	14.09	8.80	12.5	3.0	3	3900	54.0	5.4	100
12-21-50	16.49	8.90	10.5	2.5	2¾	4160	60	7.2	0
12-21-50	16.55	8.12	11.75	2.21	3	3820	51.9**	5.2	100
12-23-50	14.06	8.01	11.5	2.1	2¾	3900	71	5.2	100
12-29-50	14.33	7.59	10.0	2.0	2½	4300	67	4.5	100
1-3-51	12.45	8.61	13	1.87	3¼	3500	55	6.5	100
4-7-51	13.37	7.75	10	1.87	1¼	4300	42	4.8	80
4-7-51	14.27	7.34	10	1.87	2	4300	56	4.8	80
4-7-51	18.94	7.72	10	1.73	3	4300	92.4	4.1	80
4-7-51	13.67	7.00	10	1.83	1½	4300	43	4.9	80
4-7-51	15.50	6.60	10	2.07	2⅛	4300	68	6.9	80
4-7-51	12.20	7.70	10	2.00	2½	4300	68	5.3	80
4-7-51	15.03	7.07	10	2.00	3	4300	80	4.5	80
4-7-51	13.89	6.72	10	1.73	2	4300	61	4.4	80
8-3-51	14.73	8.86	9.5	1.89	2½	4450	74.5		75
8-7-51	14.17	9.22	9.25	2.70	2¾	4530	56		75
8-8-51	14.62	8.58	10	2.67	2¾	4300	55.5		75
8-17-51	13.84	7.53	9.25	3.0	2¾	4530	46.0		75
8-23-51	14.00	6.50	10.25	2.4	3	4230	69.0		75
9-21-51	16.3	7.48	10.5	2.75	2¾	4160	24	9.0	75
9-28-51	14.3	8.75	12.5	2.13	3	3880	38	6.0	75
							59.2 Wet weight		
Average	14.73 Pct						50.5 Dry weight		

* Based on drum speed, cake thickness, and bulk density.
** Weighed total cake discharge over time interval.

Table II. Average Size Consist of West Virginia No. 2 Gas Filter Cake, 6x10-Ft Peterson TFR Filter, Truax-Traer Coal Co., Ceredo, West Va.

U.S. Standard Mesh	Wt, Pct	Cumulative Pct On	Range—Wt, Pct
+14	6.1	6.1	3.4- 9.1
14x30	26.3	32.4	17.3-51.0
30x50	39.6	72.0	34.5-53.8
50x100	15.8	87.8	7.4-23.2
100x200	6.6	94.4	3.3-10.2
−200	5.6	100.0	3.0- 7.2

Table III. Typical Filtrate Solids Analysis, 6x10-Ft Peterson TFR Filter, Truax-Traer Coal Co., Ceredo, West Va.

	1.45 Pct Solids		2.46 Pct Solids	
U.S. Standard Mesh	Wt, Pct	Ash, Pct	Wt, Pct	Ash, Pct
+30	5.3	2.93		
30x50	9.6	2.75	3.1	
50x100	15.1	4.12	9.4	
				4.19
−100	70.0	22.90		
100x200			9.4	7.02
−200			78.1	45.76

marily −200 mesh particles, the ash content of which is very high. Loss of this fine slime material further aids in the desliming process which is largely obtained in the cyclones. The amount of loss can be regulated by the choice of slot opening for the trirod screen. Table III illustrates a typical analysis of the filtrate.

One of the important features of the Peterson TFR filter is the installation of electro-magnetic vibrators on the inside of the filter drum to create an intense vibration of the cake as it is formed and dried. A denser cake is formed which aids in squeezing out more of the water. A series of three tests at different levels of vibration were made on the same day with the units. Fig. 2 shows the reduction of 3.6 pct in final moisture content when the intensity of vibration was increased from 0 to 85 pct of full load amperage. In addition to making possible a higher moisture removal, the vibrators aid in operation by maintaining a uniform cake thickness. Furthermore, vibration appears to have a greater effect on moisture reduction when the tonnage is high, thereby effectively increasing the capacity of the filters.

The top feed reservoir feature has several advantages over conventional filter design for dewatering coal. 1—Coarse coal can be fed to the unit without a build-up in the feed reservoir. 2—The reservoir is designed especially for rapid-settling, fast-filtering products. 3—When the filter cake is formed, a more uniform size-distribution throughout the cake results than with the underfeed filter. The latter type tends to pull the finer sizes to the filter screen first, thereby reducing the filtration rate and increasing final moisture. In most cases, this deslimed material cannot be handled on underfeed filters. 4—The top feed units at Ceredo operate with a very short cake formation period, usually with a pool 1 ft deep, permitting a much longer drying time: 75 pct of the total filter area as compared to about 45 pct for the underfeed type. This gives further assistance to improved moisture percentages and tonnage rates.

It is particularly important in designing a TFR installation to provide good control and distribution of the filter feed. Proper solids to liquids ratio is essential, as well as correct size distribution. Coals with extremely high air permeability will give difficulty in priming the filter and in maintaining the cake. Locating the feed reservoir on other installations at a higher level on the drum has overcome this difficulty.

Pilot Plant Work

Laboratory tests by Peterson Filters and Engineering Co. indicated that the following variables were of prime importance in determining filtration rate and final moisture: 1—inches Hg of vacuum, 2—air rate through the cake, expressed as cu ft per min or cfm/ft² at the vacuum, 3—size consist of feed, 4—type of cake discharge from drum, 5—percent solids in feed, 6—vibration, 7—drum speed or minutes per revolution, mpr.

To present a better idea of the effect of these variables, a pilot plant unit was installed, consisting of a 10-sq ft, double-drum TFR filter supplied by the Peterson Filters and Engineering Co. The unit was fed by the underflow from a 14-in. cyclone thickener. Any proportion of the discharge from the cyclone could be bled to the filter. An oversize Roots-Connersville positive displacement blower was furnished by the manufacturer to permit any desired vacuum. The filter cake discharge was collected on a 9-in. belt and transferred to a flight conveyor carrying clean coal. This product was collected during test runs and weighed to give accurate tonnage figures. Samples for moisture analysis were taken as the coal was scraped from the drum. Table IV gives representative data for the pilot plant unit.

It was apparent as soon as operation of the unit began that the underflow from the cyclone could be thickened to the point where uniform feed to the

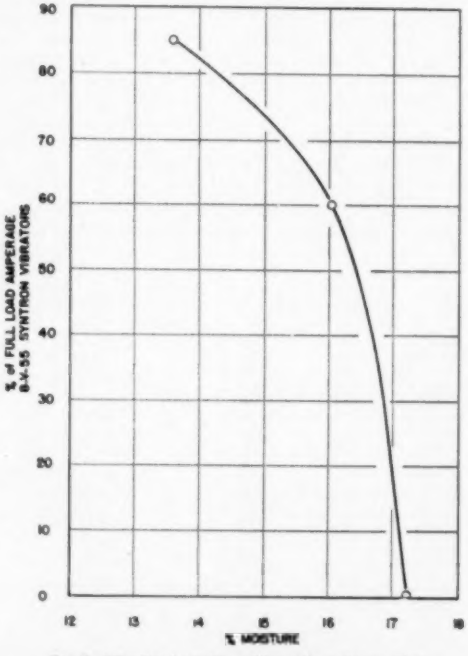

Fig. 2—Vibration intensity vs filter cake moisture. Test date, Jan. 9, 1950. Two-inch cake. Drum speed, 2 mph.

Table IV. Pilot Plant Data, 10-Sq Ft Peterson TFR Double-Drum Filter, Truax-Traer Coal Co., Ceredo, West Va.

Test No.	Filter Cake, Tons Per Hr Per 100 Sq Ft	Standard Cfm Per Sq Ft	Drum Speed, Min Per Rev	Vacuum, In. Mercury	Cake Thickness, In.	Total Cake Moisture, Pct*	Vibration	−200 Mesh in Cake, Pct
5	16.9	25.1	1.8	10.5	1.75	19.2	Off	5.3
8	39.7	15.4	1.1	13.0	2.5	26.0	Off	9.2
9	23.6	16.9	2.05	11.5	2.7	22.3	Off	9.2
10	12.9	14.6	1.8	5.1	1.3	18.9	Off	5.0
11	12.9	14.6	1.8	5.1	1.3	18.3	Off	
12	9.2	21.8	2.67	7.0	1.4	18.4	Off	4.7
13	17.1	23.7	1.8	7.0	1.8	21.5	On	
17	15.7	10.1	1.58	6.0	1.4	21.8	On	
18	15.7	10.1	1.58	6.0	1.4	23.1	Off	1.8
19	6.45	21.2	5.33	13.5	2.0	17.4	Off	4.6
20	16.2	23.0	2.07	12.75	1.9	19.9	On	
21	16.2	23.0	2.07	12.75	1.9	18.3	Off	
22	10.6	14.0	3	5.4	1.8	21.2	On	
23	10.6	14.0	3	5.4	1.8	21.0	Off	6.3
24	10.6	17.5	3	15.0	1.8	14.6	On	
25	10.6	17.5	3	15.0	1.8	19.0	Off	6.3
28	10.7	27.7	1.5	11.0	1.4	25.2	Off	14.9
29	10.7	27.7	1.5	11.0	1.4	19.1	On	
31	27.7	22.1	1.08	11.0	2.2	29.1	Off	14.9
32	5.07	28.5	3.67	10.8	1.1	19.8	On	
33	5.07	28.5	3.67	10.8	1.1	22.4	Off	
34	10.14	25.0	1.92	12.0	1.1	20.4	On	
35	10.14	25.0	1.92	12.0	1.1	23.6	Off	16.6

* Surface moisture pct equals total moisture pct — 1.8. (1.8 is U.S. Bureau of Mines published seam moisture for No. 2 gas.)

filter was impossible. This was the feature which limited thickening ahead of the filter, and it was found that maintaining feed solids concentration at about 50 pct gave best results on the pilot plant unit.

Effect of CFM/Ft², MPR, and TPH/100 Ft²

Because of the great amount of interaction between filtration variables, it is impossible to determine quantitatively the individual influences by the classic experimental method of varying only one factor at a time. Consequently it was decided to attempt a correlation based on an empirical combination of the variables. Observation of the data in Table IV suggested that the surface moisture was dependent on the variables as follows: 1—Surface moisture decreased with higher cfm/ft². 2—Surface moisture decreased with lower drum speed. 3—Surface moisture increased with higher tonnages. 4—Surface moisture increased as size-consist became smaller.

From the above, a plot of final surface moisture as a function of (cfm/ft²) (0.65 mpr)/(tph/100 ft²) was compiled as illustrated in Fig. 3. The factor 0.65 mpr was indicative of the time that air would be actually passing through the cake. A reasonable correlation appeared to exist. The significance and restrictions of the correlation will be discussed in the following paragraphs.

Test work was done on two different types of feed. Runs 1 through 27 were on material drawn from the raw coal cones through the single 14-in. cyclone. This yielded a feed similar to that expected in the full-scale operation. The remaining tests were conducted on the same type of material with the coarse fraction scalped off. Since the size-consist of the individual runs for the same feed did not vary greatly, it was decided to correlate on the basis of the average size consist or surface area. In Fig. 3 the upper curve refers to the finer coal, while the lower curve relates to the coarser material. The two curves have the same slopes and shape but are displaced from each other to give higher surface moisture for finer cakes, or higher specific area.*

* Specific area is defined as the surface area of cubes whose dimensions are the average of the dimensions of the size fractions measured in square feet per pound.

The shape of the curve is very important and indicative of the operation which can be expected. The point of sharp break is considered the location of optimum operation. To the left of this point the moisture increases rapidly. Thus overloading the filter causes a significant increase in surface moisture as insufficient air volume is drawn through the cake. To the right of the optimum, the curve is quite flat, indicating that little can be gained by drastic reduction in tonnage or overdesign of vacuum pumps. Attention should also be directed towards the necessity of obtaining a fully deslimed feed so that full efficiency of the filter can be utilized in yielding minimum moisture percent and maximum rate.

Laboratory tests indicated that substantial improvement in moisture could be expected when high frequency vibration was applied to the cake. The preliminary pilot plant data was not conclusive, as both negative and positive results were experienced. This was finally attributed to improper mounting of the vibrators in the pilot plant unit. They were bolted rigidly to the drum, which was in turn rigidly mounted to its framework. If care was taken always to sample the product at a point where vibration was a maximum, that is, over the vibrator, sub-

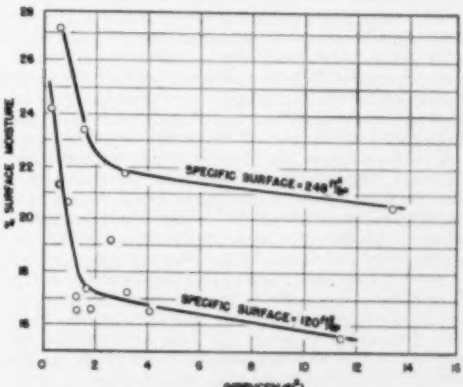

Fig. 3—Pilot plant filter tests, no vibration, showing effects of tonnage, tons per hour per 100 sq ft; air volume, cfm per square foot; drum speed, mpr; size consist; and specific surface.

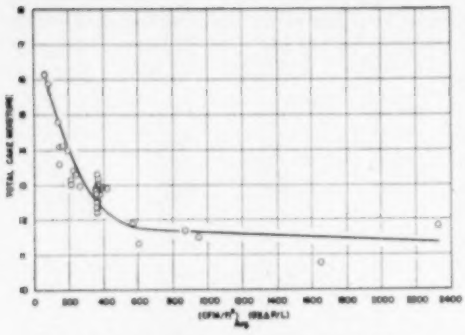

Fig. 4—Total moisture as a function of $(cfm/ft^2)_{avg}$ (θ) $(\Delta P/L)$. Experimental vacuum filtration data. No vibration. Slurry temperature, 80°F.

stantial decreases in moisture content were obtained. The reduction varied from 2.2 to 6.1 pct. Although results were not consistent, they were encouraging enough to indicate that the 6x10-ft filters should be equipped with vibrators. Therefore the filter deck on the large units was rubber-mounted to achieve proper application of the vibration. Further proof of the actual dewatering efficiency with vibration will be found in the theoretical section to follow.

One of the most important characteristics of this type of filter, soon evident in the pilot plant, was its capacity for dealing with wide fluctuations in tonnage without interruption of production. The drum speed could be rapidly controlled to adjust for tonnage changes. In addition, the Top Feed Reservoir fluid level automatically increased with the higher feed rates. This increases the pickup time and therefore cake thickness and capacity. An overflow weir is provided for extreme overload.

From the curve of Fig. 3 it was now possible to design the full-scale unit with respect to filter area and vacuum pump requirements. Accordingly the flow sheet and filters, as indicated in Fig. 1, were recommended to handle up to 100 tons of dry solids per hour.

Theoretical Vacuum Filtration Studies

To obtain a more quantitative analysis of the influence of filtration variables on capacities and final moisture contents, an experimental investigation was undertaken at Northwestern University. A small vacuum unit was constructed, similar to the one used by Peterson Filters and Engineering Co., consisting of a gasket-sealed receptacle which accommodated a 4-in. sq stainless steel B-Z tri-rod filter screen. A Syntron vibrator was also attached to the unit so that vibration intensity could be studied. All filtrates were collected in a receiver beneath the screen, and the unit was connected to a vacuum system that permitted both control of vacuum and accurate measurement of the air drawn through the filter cake. A large batch of coal from Eastern Gas and Fuel Co. was thoroughly mixed to yield a uniform size-consist throughout. This was checked by periodic wet screen analyses on different runs. An average size-consist for the experimental coal is given below in Table V.

In the performance of all tests, a prepared 60 pct solids concentration slurry was made up, thoroughly mixed, and poured onto the filter deck. The valve to the unit was opened to allow the desired vacuum, and the filtration time was recorded after the disappearance of water from the top of the cake. This would correspond to the drying arc time on the TFR filter. The vacuum was immediately broken at the end of the run and measurement was made of the cake thickness. Samples were then taken for size and moisture analysis.

It was decided that the following should be studied in an analysis of filtration variables: 1—air rate and quantity, 2—vacuum level, 3—drying time, 4—cake thickness, 5—−200 mesh content, 6—feed slurry temperature, 7—vibration intensity, 8—tri-rod screen slot opening. Accordingly an experimental program was proposed to investigate these factors.

Influence of Air Rate and Quantity, Vacuum Level, Drying Time, and Cake Thickness

It is impossible to study individually the influence of air rate and quantity, vacuum level, drying time, and cake thickness, as they are all interacting. Accordingly it was necessary to perform a large number of runs at varying conditions so that empirical correlations could be tested. All runs were made on the same type of coal at constant slurry temperature, −200 mesh content, screen slot opening ¼ mm, and zero vibration. Results are shown in Table VI.

To obtain a correlation of final moisture content as a function of these four variables, a theoretical approach is necessary. Moisture removal in vacuum filtration is primarily obtained by the drawing of air through the filter cake. As the amount of air, temperature, and contact time are relatively small, it would appear that evaporation would play an insignificant part. Rather, moisture removal will be largely a function of entrainment or propelling of water by the moving air. From these considerations, water removal or final moisture content should be primarily a function of air velocity, length of time the air passes through the cake, and cake permeability. As air velocity through the cake will increase as the air expands, it was decided to base air velocity at the average cake pressure, i.e., atmospheric pressure minus one-half the vacuum level. Air velocity through the interstices of the cake, which is impossible to measure, will be a function of air quantity at the average cake pressure per square foot of filter surface. Finally, pressure drop per unit length of cake is a function of cake permeability and accordingly can be used as a more convenient measure.

From this theoretical reasoning, a correlation for final moisture content as a function of $(cfm/ft^2)_{avg}$ (θ) $(\Delta P/L)$ was attempted where:

$(cfm/ft^2)_{avg}$ = Cubic feet of air per minute per square foot of filter area at average cake pressure.
θ = Drying time in minutes.
ΔP = Pressure drop in inches of mercury.
L = Thickness of filter cake in inches.

Table V. Average Size Consist, Experimental Vacuum Filtration Coal

U.S. Standard Mesh	Wt, Pct
+6	0.3
6x8	9.1
8x12	11.9
12x20	21.0
20x40	16.7
40x60	11.2
60x100	8.7
100x200	7.0
−200	14.1

Table VI. Experimental Vacuum Filtration Tests, Slurry Temperature 80°F, Laboratory Coal

Test No.	Vibration, Pct Full Intensity	Standard Cfm Per Sq Ft*	Vacuum, In. Mercury	Filtration Time, Min	Cake Thickness, In.	Total Cake Moisture—Wt, Pct
1	48	10.65	27.10	1.31	1.01	13.76
2	48	2.45	23.63	4.06	1.01	12.53
3	48	0.314	19.06	21.10	1.00	11.73
4	48	3.854	19.36	2.40	0.96	14.43
5	48	4.24	15.81	2.25	0.99	16.05
6	48	1.354	17.37	10.10	0.97	13.96
7	48	2.855	17.17	4.92	0.94	15.92
8	48	0.943	17.38	14.35	0.95	13.60
9	0	3.300	17.50	4.10	1.07	13.76
10	0	3.084	17.47	4.48	1.10	12.44
11	0	3.255	17.54	4.30	1.06	12.95
12	0	3.087	17.45	3.73	1.13	12.87
13	0	2.955	12.61	4.58	1.10	13.19
14	0	2.897	17.43	4.72	1.12	12.91
15	0	3.377	17.56	4.05	1.12	13.04
16	0	2.735	17.45	5.00	1.07	12.76
17	0	2.339	17.52	5.85	1.10	12.55
18	0	1.960	17.45	6.91	1.11	12.23
19	0	3.059	17.57	4.47	1.10	13.20
20	0	2.680	17.72	5.14	1.11	13.22
21	0	2.094	17.49	2.64	1.11	14.11
22	0	2.546	17.40	1.14	1.10	13.89
23	0	3.020	17.43	2.70	1.09	13.43
24	0	2.683	17.66	8.24	1.11	11.85
25	0	3.959	17.47	9.07	1.13	11.54
26	0	3.030	17.74	19.18	1.08	10.77
27	0	3.220	27.60	4.29	1.12	11.83
28	0	2.438	23.91	5.67	1.13	11.73
29	0	2.281	20.76	6.12	1.09	11.89
30	0	3.472	13.81	3.96	1.13	13.34
31	0	3.768	10.74	4.94	1.14	14.14
32	0	3.013	7.84	4.54	1.13	14.83
33	0	2.347	4.73	6.02	1.14	16.15
36	0	6.17	17.61	2.24	0.66	12.93
37	0	2.867	12.52	4.77	1.10	13.04
38	0	2.064	17.59	6.70	1.58	13.01
39	0	1.777	17.50	7.70	2.09	14.09
40	0	1.237	17.51	11.05	2.85	13.63
45	0	3.213	17.62	4.26	1.14	12.76
46	0	3.886	17.83	3.52	1.13	13.00
47	0	3.209	17.62	4.31	1.11	13.99
48	0	4.04	17.53	3.38	1.12	13.51
49	0	3.331	17.69	4.15	1.15	12.52
50	0	3.372	17.53	4.10	1.15	12.62
51	0	2.634	17.55	5.25	1.14	12.60
52	10	1.230	17.65	11.23	1.03	11.67
53	33	0.575	17.53	24.01	0.97	12.39
54	70	0.320	17.47	16.94	0.97	13.94

* Standard conditions 14.7 psia and 80°F.

Fig. 4 illustrates the reasonable correlation obtained. Other methods were also tried, but none yielded as good an agreement.

Several important concepts can be derived from the resultant curve. The decided break in the curve is again apparent as in Fig. 3, indicating the importance of designing the filter to operate at or slightly beyond this point. Overdesign merely results in wasted filter capacity or unnecessary power costs on the vacuum pumps. The important interactions of the four variables should be stressed. If cake thickness is increased too far, air rate through the cake will be reduced, resulting in insufficient air velocities to perform the potential economic moisture removal of vacuum filtration. Drying time can be increased by use of greater area, but this will automatically require greater amounts of air per square foot of area. If sufficient care is not exercised, oversizing of the filters may result.

Vibration Intensity

In order to prove the value of the correlation method, the pilot plant data of Table IV are also plotted in the same manner in Fig. 5. Again a reasonable correlation exists. Data for the pilot plant vibration tests were not plotted, as they definitely exhibited anomalies indicating that vibration was displaying a random effect. Undoubtedly, as will be later shown, this was caused by improper application of vibration.

Vibration tests on the experimental vacuum unit are also listed in Table VI. However, results were random, as with the pilot plant unit, and good agreement was not obtained. Two reasons are considered to have caused this effect. First, the filter cake in the experimental unit is formed under static conditions, and this necessitates a small pickup time, which may permit segregation to occur. With no vibration, this did not occur, but segregation did result upon application of vibration. Secondly, the high −200 mesh content, 14.1 pct, in combination with the segregation effect with vibration undoubtedly caused excessive plugging of the interstices within the cake. With lower slime feed slurries, segregation would not occur, because of the shorter pickup time; thus vibration effects could be reliably ascertained.

One quantitative indication of the effect of vibration can be obtained from the experimental data of Table VI. With equal feed slurry volumes and solids concentrations, cake thickness without vibration averaged 1.11 in., whereas with vibration this value was 0.98 in. A 12 pct decrease in cake thickness is thereby obtained by vibration, which undoubtedly aids in moisture removal when it is correctly applied.

As proof of the beneficial advantages of vibration, the operating data for the full-scale unit have been plotted in the same correlation manner in Fig. 6. The smooth curve for the pilot plant data of Fig. 5 is also indicated. It will be observed that the values lie below the smooth curve from 2 to 5 pct. The advantages of vibration become self-evident when displayed in this manner. It should also be noted that the full-scale data cluster about the correlating factor region where the break point occurs in the smooth pilot plant curve. This indicates that the filter has been correctly scaled-up for the installation.

Feed Slurry Temperature

As filter cake temperature increases, air velocity through the cake will also increase at the same standard cfm/ft² rate because of the temperature expansion effect on the air. Further, viscosity and surface tension of the water will also decrease, thus improving the permeability of the cake. Theoretically, therefore, feed temperature should have a quantitative effect on final moisture content.

A series of tests were made at varying feed temperatures as reported in Table VII. Comparing final water contents of the runs, it is apparent that moisture definitely decreases with an increase in temperature. Furthermore, it will be observed that all runs were made at essentially the same correlating coefficient, offering proof of temperature influence.

Table VII. Experimental Vacuum Filtration Tests, Varying Feed Slurry Temperatures, Laboratory Coal

Slurry, Degrees F	Standard Cfm Per Sq Ft	Vacuum, In. Mercury	Filtration Time, Min	Cake Thickness, In.	Total Cake Moisture—Wt, Pct	$(Cfm/Ft^2)_{STD}$ (θ) (ΔP/L)
40	2.787	17.66	4.96	1.09	13.34	384
58	2.620	17.55	5.22	1.10	12.43	372
100	2.680	17.77	5.16	1.14	12.51	373
120	2.520	17.60	5.43	1.13	12.13	384

C. Fuhrmeister in an earlier paper[1] listed industrial data for vacuum filtration indicating an increase in filtration rate, expressed as pounds of cake per square foot per hour, of 1 pct per degree Fahrenheit. Furthermore, he observed a 1.6 pct drop in final moisture content for an 80°F temperature increase which closely parallels this work.

Mr. R. L. Sutherland of the Truax-Traer Coal Co. has noted[2] that there is a close correlation between final moisture content of the finished coal at the Ceredo plant and the circulating water temperature. In the winter months when water temperatures are approximately 40°F, average moisture content of the loaded coal is 5.8 pct. When water temperatures reach 79°F in the summer months, moisture content is only 4.75 pct. This points to the possible advantages of a closed water system wherein water temperatures can be maintained at 70° to 80°F to obtain approximately a 1 pct decrease in final moisture contents. When thermal drying installation and operating costs are considered, it may be more economical to maintain water temperatures at 70° to 80°F and blend the filter cake with the coarse coal. Studies are worthwhile to ascertain not only how much water would be removed by the two methods, but also exactly how much each pound of water removal costs.

Tri-Rod Slot Openings

Referring to Table IV, runs 46 and 50 were made with 0.001-in. B-Z tri-rod screen, runs 45 and 49 with 0.005-in. B-Z tri-rod screen, and runs 47 and 48 with 0.020-in. B-Z tri-rod screen. The remainder were all made with 0.010-in. B-Z tri-rod. No appreciable effect on the final moisture content could be observed. However, there was a decided influence, as would be expected, upon the loss of solids in the filtrate. Table VIII gives a summary of these effects; special attention should be paid to the −200 mesh removal.

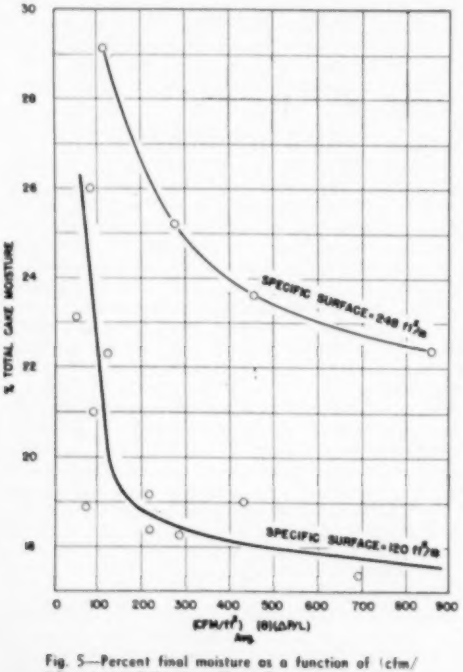

Fig. 5—Percent final moisture as a function of $(cfm/ft^2)_{Avg}(\theta)(\Delta P/L)$, from Truax-Traer pilot plant data. No vibration applied.

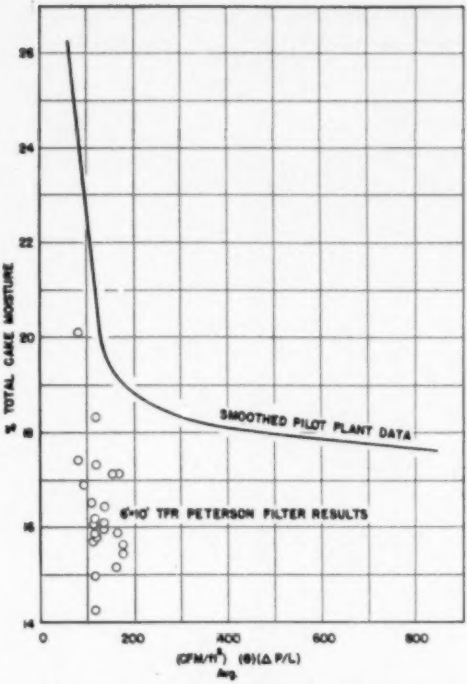

Fig. 6—Comparison of 6x10-ft TFR Peterson Filter results with vibration and smoothed pilot plant data without vibration.

It will be observed that slime removal increases steadily with opening. However, loss of desirable solids is probably excessive for the 0.020-in. size. Thus it is apparent that appreciable removal of high ash slimes can be effected in the filter but will be limited to the coarse solids loss that can be tolerated. However, 10.3 pct slime removal obtained with the 0.010-in. screen would represent a desirable decrease in final ash content of the filter cake. Naturally, the percentage amount of slime removed will also vary with cake thickness.

−200 Mesh Content

The influence of slimes upon final moisture content and filtration capacity has been stressed in many articles. Undoubtedly this is a prime factor in filter operation. Most authors emphasize the importance of desliming in increasing filtration rates and moisture removal. Fuhrmeister indicated data in his paper where a 25 pct increase in −200 mesh content decreased filter cake tonnage rate 30 pct and increased moisture content 38 pct. The economics of such unwanted inclusions are self-evident. Similar evidence is presented in the two curves of Fig. 3, where appreciable increases in moisture content and lowered rates were experienced with higher slime content coal.

A series of runs were made on the experimental coal with varying amounts of −200 mesh pct. Results are given in Table IX. Upon investigation of the data, it will be apparent that filtration time decreases rapidly with a decrease in −200 mesh content, indicating the sizable increase in filtration capacity that results. Furthermore, there is a definite decrease in final moisture content, although not as marked. It should be noted that air rate becomes excessive at slime contents below 2 pct. This is definite proof of the serious influence of slime percent on cake permeability. However, it should be apparent that some slime content is probably necessary for vacuum filtration. If the cake becomes too permeable, it will be difficult to prime the filter and hold a cake. As previously mentioned, this can be overcome by modification of filter design.

Filtration installation and operation costs are high, while desliming costs due to the advent of the cyclone are very low. Thus the economics certainly justify the inclusion of desliming equipment when necessary in any filtration flowsheet.

Filter Scale-Up

To prove that filters can be scaled-up by the experimental tests and correlation methods outlined in this paper, runs were made on the vacuum unit at Northwestern on sample material from the feed to the full-scale units at Ceredo. Complete operating data and moisture tests were recorded at the plant on the day each sample was taken. The correlation between the tests is indicated in Table X. Excellent agreement was obtained, as attested by the similarity of the data. It is feasible, therefore, first to run simple filtration tests, taking the recommended data, and thus scale up the unit so that operation will result at maximum economy.

Summary

Successful application of the Peterson Top Feed Reservoir vacuum filter to mechanical dewatering of 14x150 mesh coal has been achieved at the Truax-Traer Coal Co., Ceredo, West Va. Surface moistures will range from 13 to 17 pct, and capacities up to 100 tons per hr of dry solids have been handled on the two 6x10-ft units. Maintenance and repair have been negligible, while operation is uniform and relatively simple, in spite of the large surges in solids rate of this fine coal. Vibrators are also utilized on the filters where they serve to compact the resultant smoother cake, and thus squeeze out more of the water. The pilot plant studies to obtain feasibility and scale-up information are discussed and special emphasis is placed on correct full-scale design. Care must be taken so that filtration equipment is not overdesigned, causing excessive power costs and inefficiently used capacity.

An experimental program on vacuum filtration investigated all important variables under closely controlled conditions. A reasonable correlation for final moisture as a function of air rate, drying time, and cake permeability ($\Delta P/L$) for a specific coal was obtained, which also appeared theoretically sound. This method can be used on any fine solids for filter scale-up by making a few simple vacuum filtration tests as indicated with such data being measured.

Vibration was proved to be beneficial for the Truax-Traer installation, as it lowers final moisture by 2 to 5 pct. Tri-rod screen size appeared to have little effect on total water content, although small but important desliming was achieved with little loss of +200 mesh coal up to 0.010-in. slot opening. Slurry feed temperature increase will decrease water content as well as increase filtration rate. Special studies should be made of the economics of heating circulating water as against the high expense of thermal drying. Finally, the importance of desliming has been stressed, as slime inclusions exert a pronounced effect by raising moisture and lowering filtration capacities. In most applications of vacuum filtration to coal solids, desliming will be mandatory to eliminate this troublesome material, as economics will more than justify it.

Table VIII. Influence of Slot Opening on Slime Removal and Solids Loss

B-Z Tri-Rod, Nominal In. Slot Opening	Feed Solids in Filtrate, Pct	−200 Mesh in Filtrate Solids, Pct	Loss +200 Mesh to Filtrate, Pct*	−200 Mesh Removal in Filtrate, Pct
0.001	0.80	70.4	0.28	4.0
0.005	1.75	61.1	0.79	7.6
0.010	2.50	58.1	1.22	10.3
0.020	5.60	46.4	3.50	18.4

* Based on 14.1 pct −200 mesh in feed.

Table IX. Influence of −200 Mesh Content on Filtration Results, Laboratory Experimental Coal

−200 Mesh, Pct	Standard Cfm Per Sq Ft	Vacuum, In. Mercury	Filtration Time, Min	Cake Thickness, In.	Total Cake Moisture—Wt, Pct
14.1	3.020	17.43	2.70	1.09	13.43
7.7	8.91	17.65	1.55	1.13	13.00
5.1	13.82	17.50	0.98	1.07	13.36
1.8	115	7.60	0.24	1.11	11.75
0.7	251	17.08	0.11	1.07	9.47

Table X. Comparison of Actual and Experimentally Predicted Results

	Test 1		Test 2	
	Actual	Experimental	Actual	Experimental
Total cake moisture, pct	18.1	18.85	16.1	16.31
Standard cfm/ft²	17.1	17.24	14.8	20.63
Vacuum, in. mercury	10.5	10.7	12.5	12.45
Cake thickness, in.	2.5	2.27	3.0	2.80
Drying time, min.	1.85	1.81	1.39	1.39

Acknowledgments

The authors wish to thank the Truax-Traer Coal Co. and its personnel for their consideration and constant cooperation. Special appreciation is due Mr. R. L. Sutherland of that company for his help, thoughtfulness, and valuable advice.

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Features Incorporated in the Design of Lone Star's Two New Dry Process Cement Plants

by Claiborne C. Van Zandt

THE heavy postwar demand for Portland Cement has created shortages that are gradually being overcome by increases in plant capacity. In the postwar period, the Lone Star Cement Corp. has expanded several of its existing 15 plants and has built 2 new plants, now in operation, with a third new plant under construction in Brazil, of design somewhat different from that of the two preceding ones.

The new domestic plants are located at Lone Star, Va., near Roanoke, and at Maryneal, Tex., near Sweetwater, and went into operation during 1951. Each plant is equipped with two rotary kilns and has an initial capacity of 1,500,000 bbl per year, or 6,000,000 sacks. The total cost of each plant was about $10,500,000, and on this basis $7.00 in 1951 money are required per bbl of annual capacity; at 376 lb per bbl, this corresponds to $37.00 per ton of output. The investment corresponds to approximately $80,000 per hourly-wage earner employed.

The purpose of this paper is to describe the most interesting design features of the two new and almost identical Lone Star plants.

Why the Dry Process Was Chosen

The two new domestic Lone Star mills employ the dry process because: 1—The raw materials at both properties are particularly suitable and can be dried very readily. 2—The Maryneal, Tex., plant does not have water available for slurry. 3—Very substantial kiln fuel savings can be obtained. 4—Dry raw grinding reduces ball and liner wear and general maintenance, without any sacrifice in power.

Operating costs to date have more than justified the choice of the dry process for these two plants.

Raw Materials

The chemical composition of Portland Cement raw material after crushing, grinding, and mixing, but before burning, is about as follows:

Item	Pct	
$CaCO_3$	75-78	Limestone, marl, and shell
$MgCO_3$	1-3	
SiO_2	13-15	Shale, clay, and blast furnace slag
Al_2O_3	3-5	
Fe_2O_3	2-4	

Shales and clays generally do not contain sufficient SiO_2 and Fe_2O_3, and these components must, therefore, be added in the form of sand, iron ore, and sometimes tailings from an iron ore concentrating plant. Certain objectionable materials occur in raw mix: magnesia, limited in cement to 5 pct, in excessive amounts; alkalies, as sodium and potassium; phosphorous; and sulphur, generally present in coal and oil.

The Roanoke raw materials are dense limestone and shale. The Maryneal materials are soft limestone and a hard brittle clay that can be handled much like shale. Sand and iron ore are required in small quantities.

General Plant Layout and Construction

The two plants are so similar that one joint description will cover both, with such exceptions as will be especially indicated.

The crushing plant is located in the quarry, and raw material is conveyed by belt to the storage building, where all materials are stored and handled. Fig. 1 identifies the various structures of the Roanoke plant. The large storage building is used for the storing and handling of all materials, including limestone, shale, sand, iron ore, coal, clinker, and gypsum. Directly adjoining the storage building is a structure under a similar arched roof, containing the raw and clinker grinding departments and the electrical substation. The discharge ends of kilns and coolers, and coal or gas burning equipment are also placed at one end of the mill building. The 340-ft long kilns extend away from the main buildings to form the long leg of an L, at the end of which is the feeding and kiln dust collecting equipment, and the raw blending silos are conveniently located adjacent to the feed end of the kilns.

The machine shop, storeroom, electrical shop, foremen's office, service rooms, and meeting hall are combined in one structure in the center of the plant, and the office and laboratory are close by.

The cement silos and packhouse do not have to be tied directly into the manufacturing process and are therefore located at a point convenient to the railroad tracks.

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The compact arrangement with centralized service, shop, offices, and laboratory has greatly simplified maintenance and supervision. Full provision has been made within the existing buildings to install additional equipment for increasing plant capacity at Roanoke by 50 pct and at Maryneal by 100 pct. Electrical power systems, pipelines, conveyors, and shipping facilities are also installed for these higher ultimate capacities.

Operations are on a 24-hr, 365-day basis except at packhouse, quarry, and crushing plant, where a 40-hr work week is normally observed.

Almost the entire plant is constructed of reinforced concrete, including the two continuous barrel roofs covering the storage, burning, and grinding buildings. Other concrete or portland cement products are also employed, such as concrete block and Transite siding and roofing. Aside from the use of the company's product, it is found that concrete construction results in a much more attractive structure, which is free of vibration and remains much cleaner than a fabricated steel structure. Painting is required only on machinery items. Corrulux translucent siding has been widely employed at Maryneal with very satisfactory results, building interiors being very well lighted during the daytime.

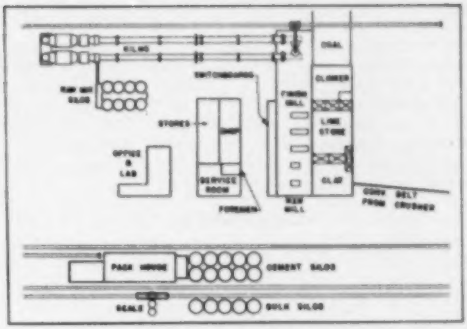

Fig. 1—General plant layout, Roanoke.

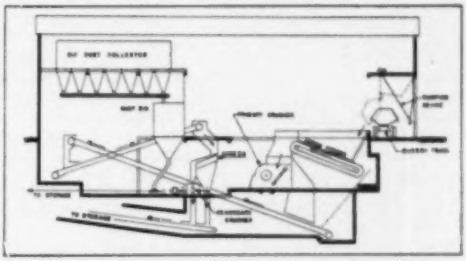

Fig. 2—The crushing plant.

Quarrying and Crushing

Bucyrus No. 27-T well drills are used for blast holes at Roanoke, and limestone and shale are removed from the quarry by two Marion 3 cu yd type 492 electric shovels and Mack tractors equipped with Easton 16-ton side-dump trailers.

Primary crushing is done at Roanoke in a 42-in. Allis-Chalmers gyratory driven by a 200-hp motor followed by a CF-15-50 Pennsylvania reversible impactor driven at 900 rpm by a 400-hp motor. The impactor is operated in closed circuit with two 5 ft 0 in. x 11 ft 6 in. rod-deck Ripl-Flo screens with rod spacing of ⅝ in. At the Sylvester quarry, about 35 miles from Maryneal, a hard clay resembling shale is dug with a 1½-yd Marion diesel drag line and transported by rail to the plant, where it is crushed in a Pennsylvania roll crusher. At Maryneal a Joy Model 225 rotary drill is mounted on a Ford chassis and is used for drilling both the limestone deposit at the plant and the clay deposit at Sylvester. A 4-cu yd Bucyrus No. 100-B electric shovel and Mack tractors with 16-ton Easton trailers supply limestone to a Pioneer 72-in. x 21 ft 6 in. apron feeder.

Fig. 3 is a diagram of the crushing plant at Maryneal. The primary crusher, a No. 7260 Dixie Hammermill, is followed by a 6 ft x 12 ft Ripl-Flo scalping screen and a No. HM-05 Hammermills, Inc. crusher for oversize.

An Eriez Metal Detector is installed at the belt conveyor leading from the primary to the secondary crusher. It consists of a detector coil which circles the belt, and an electronic control cabinet. When a

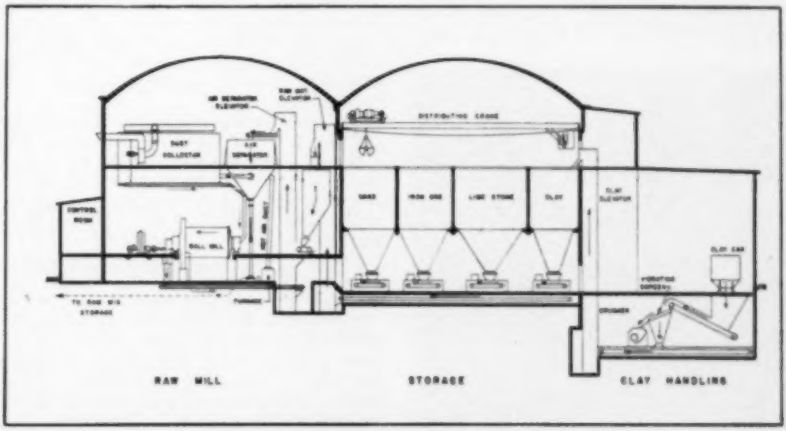

Fig. 3—Raw grinding and clay handling system.

piece of tramp metal, magnetic or nonmagnetic, passes through the coil, the belt and primary feeder are stopped and a horn is blown. The operator then knows the spot where he will find the tramp metal, and after its removal he restarts the equipment. This device and a similar detector which has been in use for about 18 months at a Lone Star crushing plant in Alabama have resulted in the discovery and removal of much objectionable metal that would otherwise have caused extensive damage.

Storage Buildings

The storage building is a reinforced concrete structure 75 ft wide and 340 ft long, served by two 10-ton Bedford cranes with 3-cu yd Blaw-Knox clam shell buckets. The various raw material components and cement clinker are stored where designated by the Chief Chemist and are transferred to the feed bins as required. A Merrick Feed-O-Weight is located under each feed bin, so that materials can be accurately withdrawn at a predetermined rate.

The thin concrete barrel roofs represent a type of construction rapidly gaining favor. They are economical, require only a small movable form, and present a very cleancut, pleasing appearance.

Drying and Raw Grinding

Fig. 3 shows the flowsheet of the raw material drying and grinding system, as well as the clay crushing plant installed only at Maryneal.

Raw materials are delivered by bucket crane to the raw feed hoppers. Merrick Feed-O-Weights measure out each material in the required proportions and the mixture is then delivered to the raw ball-mill feed bins. From here it is withdrawn as required; the feed rate is manually controlled by the operator who occasionally checks the recording ammeter connected to the air separator elevator motors. A circulating load of 500 pct is maintained.

Two completely separate raw grinding units are employed. Each unit is composed of a 9½-ft diam by 15-ft single compartment Allis-Chalmers ball mill driven by a 700 hp motor, operated in closed circuit with a 16-ft Sturtevant air separator driven by a 100-hp motor. A mill with motor is shown in Fig. 4. The raw material is fed directly to the air separators, which are supplied with hot air for drying by means of oil or gas fired air heaters. Each air heater automatically supplies the amount of heat required to drive off the moisture. These

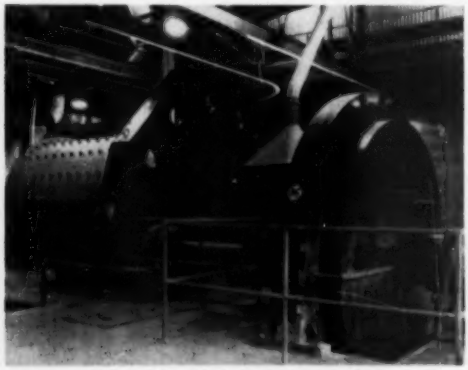

Fig. 4—Allis-Chalmers ball mill and motor.

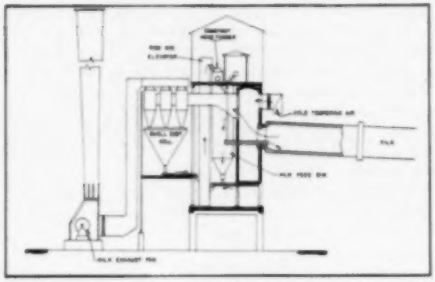

Fig. 5—Kiln feed arrangement.

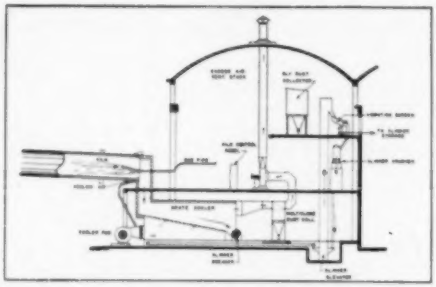

Fig. 6—Kiln department, burning and cooling system.

heaters are fired with oil at the Roanoke plant and natural gas at Maryneal. Each air separator discharges tailings to its respective ball mill, the mill output being returned to the air separator. The moisture-laden gases leaving the air separator are held to a maximum temperature of 200°F and are exhausted through a 35-ft long, 18,000 cfm Northern Blower dust collector and fan driven by a 50-hp motor. No hot air is passed through the mills, as all the drying is done in the air separators. Separators, duct work, conveyors, and elevators are all thoroughly insulated to retain as much heat as possible in the system and prevent condensation. The heat of grinding within the ball mills is thus maintained and assists materially in the drying. It is believed that these raw grinding and drying installations are the first to employ ball mills in closed circuit having air separators with raw material fed first to the separators, and the results have far exceeded expectations.

The actual performance data at Roanoke is given in Table I. The figures represent a monthly average and are not for test periods. On some days the output greatly exceeds the overall average. The output is much higher than required and the mills have never been fully charged to determine their ultimate capacity.

The raw grinding system operates without trouble, even though at Maryneal the moisture content of the feed has averaged 5.9 pct, with periods of 8 pct.

The heat delivered to the separators is adjusted automatically to take care of moisture fluctuations in the raw material. Approximately 15,000 cfm of hot gases are drawn through each Sturtevant separator. Exit temperature is kept at 180° to 200°F, while the inlet gases from the air heater will vary from 400° to 850°F maximum. Moisture-laden air

Fig. 7—Air quenching cooler.

from the separators is exhausted through a collector. The separator fines run less than 1 pct moisture, and are conveyed by a Fuller-Kinyon system to the raw blending silos located near the feed end of the kilns. Every part of each raw grinding system is under suction and is connected to the Norblo unit which handles the gases from the air separator. The raw mill department is therefore entirely dust-free.

Table I. Typical Data for Two Raw Grinding Units Operated in Parallel, Lone Star Plant, Roanoke, Va., and one Unit, Lone Star Plant, Maryneal, Texas

ROANOKE, VA.	
Output, monthly average	70 to 80 net tons per hr
Fineness	91 to 93 — 200 mesh
Moisture in feed, 30-day average	Dry months 2.6 pct Wet months 4.3 pct
Moisture in product, 30-day average	.15 to .30 pct
Typical Power Requirements	**KW**
Two 9½x15 ft ball mills	1000
Two 16-ft air separators	129.6
Two air separator elevators	33.8
Two Norblo exhaust fans	48.6
Two oil furnaces	18.2
All feeders and conveyors	41.7
	1271.9
Fuller-Kinyon system raw mill to silos	139.1
Kw-hr per ton mills only	12.5
Kw-hr per ton auxiliaries	3.4
	15.9 kw-hr per ton
Requirement for the Fuller-Kinyon system	1.74 kw-hr per ton
Heat Requirements	
Fuel oil per ton	.55 to .85 gal
Btu per ton of material dried	80,000 to 131,000
Btu per bbl of equivalent cement	23,000 to 38,000
Gross heat required per lb water evaporated	1700 Btu
MARYNEAL, TEXAS	
Moisture in feed, 8-month avg	5.9 pct
Moisture in feed, 30-day avg	8.0 pct (in wet months)
Moisture in product	.65 pct
Gross heat required per lb water evaporated	1730 btu

Raw Mixing and Blending

The blending system includes 11 silos 85 ft high, each with a capacity equivalent to 3000 bbl of cement. A Fuller rotary feeder is placed under each silo discharge cone, and each feeder has an individual adjustable speed drive. The conveying system permits blending, re-blending, and transferring, as required. A passenger elevator connects the various working floors.

Table II. Typical Kiln and Cooler Operating Data, Lone Star Plant, Roanoke, Va.

Kiln output, monthly avg	2000 bbl each
Coal consumption, as received	70 lb per bbl
Heat consumption at 13,400 btu	954,000 btu
Kiln exit gas temperature	1200° to 1400°F
Temperature of exit gas, cold air mixture	650°F
Kiln speed, actual	40 to 50 rph
Kiln motor input, normal	25 kw
Coal fineness	87 pct — 200 mesh

Fig. 5 shows the kiln feed and equipment. Kilns are 10 ft in diam and 340 ft long and are placed on a slope of 7/16 in. per ft. The refractory lining is 6 in. thick. The actual power normally required to drive a kiln at 50 rph is 25 hp. The main gears encircling the kilns, of Falk single helical design, are driven by matching pinions which in turn are driven by Falk herringbone speed reducers. There is no vibration at any kiln speed such as frequently occurs with kilns equipped with straight spur gearing. The kilns are equipped with auxiliary gasoline engine drives for use in the event of power failure; these have been used many times.

Burning and Cooling

Each kiln is nominally rated at 2000 bbl output per day, and the burning zone temperature is about 2700°F, see Table II. They are fired at the lower end, the fuel being coal in Virginia and natural gas in Texas. Feeders are synchronized with drives to maintain a constant bed depth within the kilns.

Raw material is fed into the upper end of the kilns with horizontal screw conveyors, and the short drop from screw to kiln lining results in little dusting. This method is recommended as a great improvement over the conventional long inclined feed spout so often used. Fine raw mix drops from such a spout with a splash, and much material is then airswept out of the kiln.

The kiln exit gases pass through a Buell dust collector and then through a Prat-Daniel fan. At the Roanoke plant, the collector is followed by a Koppers Elex electrical dust precipitator which collects the very fine fractions. All dust is returned to the feeding system and thus back to the kilns.

Since the dust collectors and fans are constructed of ordinary steel, a maximum gas temperature of 700°F must not be exceeded. This is provided for by mixing cold air with 1300°F gases from the kiln.

Fig. 6 shows the cooling and burning arrangement at the discharge end of the kiln. The hot clinker dis-

Fig. 8—Kiln control panel.

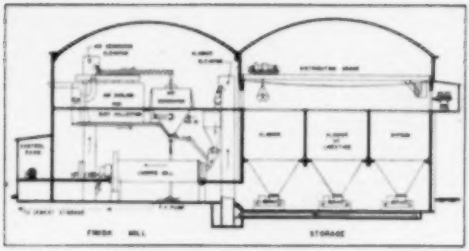

Fig. 9—Clinker grinding system.

charges to a Fuller grate-type air quenching cooler. Cold air is blown up through the clinker grates, and the hottest portion of the resulting heated air is returned to the kilns as combustion air. In order to bring the temperature of the clinker down to approximately 150°F, it is necessary to use considerably more air than is required for combustion. Excess air is vented through Multiclone dust collectors and fans, at a temperature varying from 200° to 300°F. The dust collectors recover the coarse clinker dust that would otherwise accumulate on the roofs and adjacent machinery.

Fig. 6 also shows an interesting feature developed by the Lone Star Cement Corp. now used on all grate-type cooler installations at company plants. It consists of a small hammermill which crushes all +1½-in. material and returns it to the inside of the cooler. Approximately 7 hp is consumed in this crushing operation. Since the oversize clinker is hot, it is given a second opportunity to cool when thrown back upon the grate.

Hammer wear is so slight that the clinker breakers need only be given attention once a year.

Although the clinker breakers reduce the material to −1½ in. in size and insure complete cooling by breaking up the hot chunks, much of the material is nevertheless too large for efficient grinding. All clinker is therefore sent over Aero-Vibe screens and the tailings recrushed in a No. 330 Hydrocone crusher to −⅜ in.

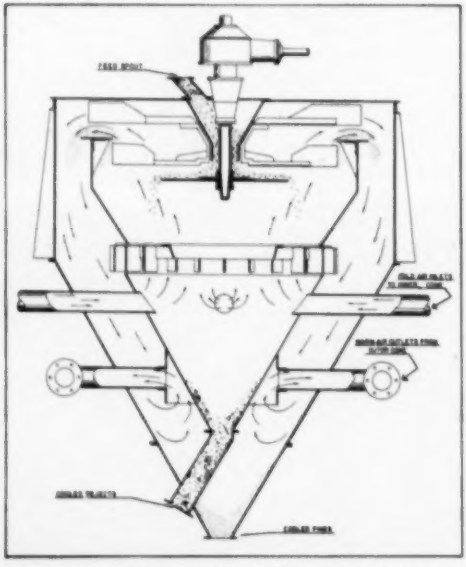

Fig. 10—Air separator arranged for Norblo cement cooling system.

Fuel Firing

Raymond bowl mills serve as unit pulverizers for each of the Roanoke kilns. The coal mills are placed in the cooler rooms so that coal feeders can be on the burning platform where they can be watched by the kiln operator. The Maryneal natural gas burning system involves a 15-in. gas manifold automatically kept under 35-lb pressure. Burners are 3-in. pipes with 2-in. tips, and the tip pressure is about 5 lbs.

The kiln control panel illustrated on Fig. 8 carries instruments affecting the operation of the kilns and coolers, and a few essential controls that have been found by experience to be very useful. These include kiln exit gas temperature recorder, mixed kiln exit gas-air temperature recorder and warning horn, kiln speed recorder, natural gas flow meter, or coal feed controller, damper regulators, draft gages, feeder and kiln revolution counters, overgrate cooler pressure control, cooler grate speed control, kiln speed controller, and ammeters.

Clinker Grinding

Fig. 9 shows the clinker grinding department. The three main feed bins are placed within the storage building, one being reserved for clinker, one for limestone, and one for gypsum. Materials are drawn from the bins by means of totally enclosed Feed-O-Weights discharging into a drag conveyor, which in turn delivers the material to an elevator discharging to a second drag which fills the individual bins feeding each mill.

To prevent segregation, the individual mill bins are small, and individual feeders deliver feed to the mills at a rate determined by the power consumption of the elevator feeding the air separator. If the circulating load, which normally amounts to, say, 300 pct, increases or decreases below the desired value, a corresponding response appears on a recording ammeter and an adjustment is made manually by the operator.

Table IV. Typical Operating Data for Three 8x32-Ft Clinker Mills Operated in Parallel, in Closed Circuit with 16-Ft Air Separators*

Net output per mill	112 bbl per hr
Cement fineness	95 to 96 pct −325 mesh
Cement surface area	1750 to 1800 Wagner
Circulating load	About 400 pct
Mill motor input	640 Kw
Kw-hr per bbl, mill only	5.7
Kw-hr per bbl, auxiliaries	1.1
	6.8 kw-hr
Cement temperature	150°F

* Power figures do not include Fuller-Kinyon system.

The grinding equipment consists of three 8x32-ft two-compartment Allis-Chalmers Compeb Mills equipped with Lorain liners and operated in closed circuit with 16-ft air separators. Each mill is driven by an 800-hp motor, and each air separator by a 100-hp motor.

Each grinding unit is equipped with a 30-ft dust collector which keeps the entire grinding operation under suction and also draws about 10,000 cfm through each air separator to cool the cement. Fig. 10 illustrates the general method by which cold air

is drawn into the inner cone of the separator and hot air removed from the space between the inner and outer cones. Table IV gives the actual operating performance. This system of cooling is very effective in that it cools the tailings as well as the finished cement and thus maintains the temperature of the grinding mill much lower than would otherwise be the case.

Cement ground without air cooling is generally discharged at a temperature between 230° and 270°F, sometimes higher. With the Norblo cooling system pictured, the temperature of the cement runs about 140° to 160°F as it leaves the grinding system, and cools considerably more by the time it reaches the packhouse. One test shows that 64 pct of the mechanical energy consumed by a Compeb mill is ultimately recovered as heat from the cooling air.

Ten clinker grinding units operated by Lone Star are now aircooled in this manner. This system is a great improvement over the conventional method of spraying the outside of the mill shells with water. It is interesting to note that the cement cooling system and the raw material drying system are alike, hot air being supplied to the air separator for drying and cold air for cooling.

Cement Storage

One group of silos is set aside for cement to be packaged, a second group exclusively for bulk cement.

Through the use of Fuller-Kinyon pumps, piping, valves, and hose switches, complete flexibility is maintained and cement from any silo can be delivered to any packing point. A passenger elevator serves the silos.

Bulk and Package Shipments

As about 65 pct of all cement is now shipped in bulk, one set of silos was set aside solely for this purpose, and cement is loaded directly from the side of each silo as shown in Fig. 11. A special bulk loading station and track scale are also provided, and cement for any silo can be pumped to this point. Fig. 11 shows a cross-section of the packhouse, where four packing stations are installed. One station is set aside solely for Masonry cement, and since there is no interconnection of equipment, contamination cannot take place. Another packing station is set aside for Incor, either air-entraining or plain. The two remaining stations are reserved for various types of standard cement. A unique feature of all the Lone Star packing plants is the circulating system employed. A constant head is maintained over the packing machines at all times, resulting in uniform and very accurate weights. Circulating tanks are provided so that a quick switch can be made from one type of cement to another.

The packing plants are kept immaculately clean, and working conditions are attractive.

General Dust Suppression

Many engineers have expressed surprise at the extensive use of drag conveyors in these new plants. Drags are often considered outmoded equipment never to be used where belt conveyors could be employed. There is no question about the low cost of belt conveyor operation, and this has been used where warranted. Belt conveyors, on the other hand, are dusty, and being a constant source of spill, they require continual cleanup. Properly designed drag conveyors are entirely self-contained and com-

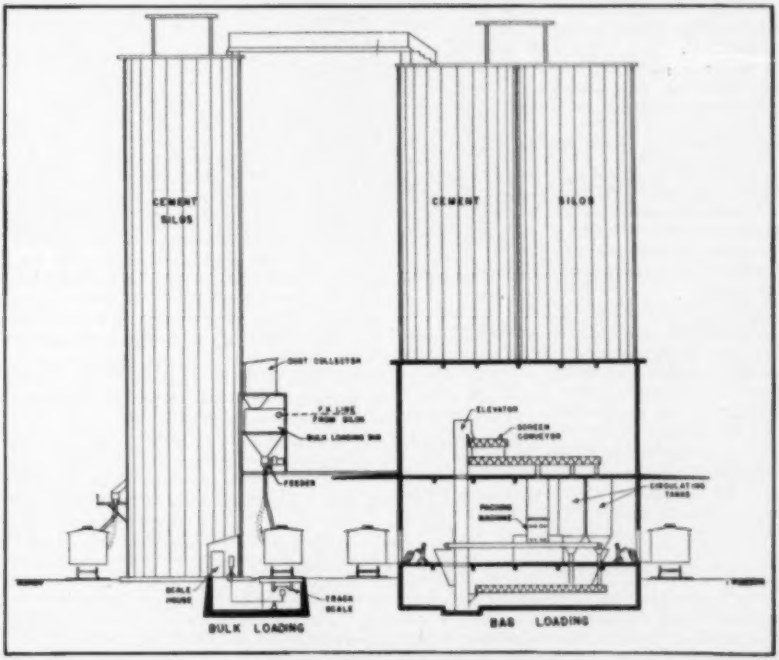

Fig. 11—Cement storage and shipping department.

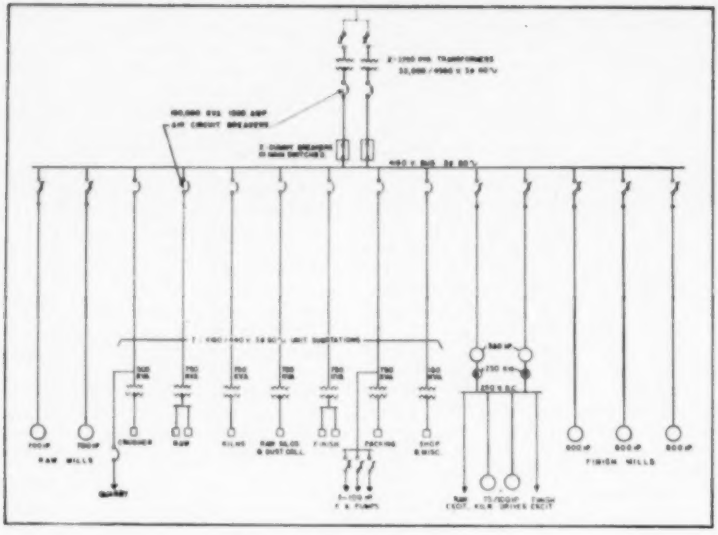

Fig. 12—Electrical distribution system.

pletely dustless. They require no cleanup whatsoever. The essential wearing parts are made of manganese steel, and maintenance is very moderate. All the drag conveyors have tight covers, and one small dust pipe connection serves to keep an entire drag system clean.

Every machine, hopper, and silo in each plant is kept under suction, and this feature, together with good housekeeping, results in a well-swept appearance. The most serious dust problem at Maryneal is to keep the buildings tight enough to prevent dust entering from without during occasional dust storms.

Shops, Offices, Storeroom, and Laboratory

A concrete building 82x182 ft long is used for machine shop, storeroom, electrical shop, foremen's office, locker rooms, and washrooms. On the second floor of the building there is also a room that is used for safety meetings and for many other purposes. A serving kitchen with stove and refrigerating equipment is provided.

The office and laboratory is an L-shaped concrete building, completely air-conditioned, with superintendent's office in the center of the L, the assistant superintendent's office and drafting room at one side, and a meeting room with special ventilating equipment at the other. Foremen's meetings are held every morning, and the room also serves to take care of special meetings which frequently occur. The remainder of one wing of the L houses the laboratory, and the other wing takes care of the clerical and timekeeping office. Washroom and locker room facilities are centrally located.

Electrical System

Fig. 12 is a one-line diagram prepared for the Roanoke plant. The electrical shop is located at one end of the machine shop, in the most central part of the plant. Typical power requirements at Roanoke are as follows:

Item	Pct
Quarry	.28
Crushing	.65
Raw drying and grinding	5.70
Blending	1.20
Burning and cooling, and kiln dust collecting	3.90
Coal burning*	1.07
Clinker storage and grinding	8.10
Packing and shipping	.58
Total	21.48 kw-hr per bbl

* The Maryneal, Tex., plant has no coal burning charge, as gas is used for fuel.

A 4360-v distribution system supplies all the General Electric unit substations, the quarry shovels, and all motors 300-hp and larger, including Ball and Compeb Mill synchronous motors and synchronous M-G sets. The synchronous motors are 80 pct power factor machines and the system power factor is kept near unity.

All motors 300-hp and larger are wound for 4000/2300 volts so they can be transferred to any Lone Star plant operating at 2300 volts if the need should arise. These large motors and the main switchboard were supplied by Allis-Chalmers. All motors 250-hp and smaller and all gearmotors were supplied by General Electric. Each mill motor is equipped with inching equipment used for spotting the mills for inspection or charging.

The main switchboard, all synchronous motor starters, M-G sets, raw and finish unit substation, raw and finish grinding multi-unit control centers, and D.C. distribution equipment are placed in the substation lean-to adjacent to the mill building. A General Electric 4160/480-v unit substation is installed in each department of the plant to handle the 440-v load. All main circuit breakers are rated at 150,000-kva interrupting capacity and 1200 amp.

The main substation and where possible other substations are supplied with filtered air to eliminate the costly maintenance resulting from dust and moisture.

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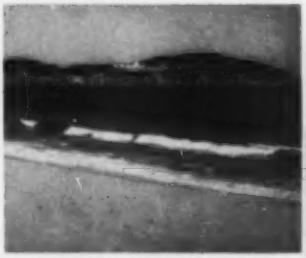
C. M. Cooley, Mining Branch Secretary

aime NEWS

Nova Scotia Prepares Welcome Mat for Industrial Minerals Division

The Industrial Minerals Div., of the AIME meeting scheduled for Nova Scotia Sept. 8-12, 1953, promises to be one of the outstanding gatherings ever attended by the group. Complementing the educational value of technical sessions will be the superb fishing, boating, golfing, and tennis facilities.

Sponsor of the meeting is the Nova Scotia Mining Society. The Canadian Institute of Mining and Metallurgy and Industrial Minerals Div., AIME have been invited to participate. The land immortalized by Longfellow's poem, *Evangeline*, still contains much of the forest primeval—enough to make it a wonderful vacation spot for delegates and their families. Its scenic beauty is almost unmatched in the area north of the border. Quoting from the Nova Scotia tour book; "Nova Scotia, once known as Acadia, is 374 miles long, from 600 to 100 miles wide . . . Excepting the Isthmus of Chignecto, 17.5 miles wide, the province is entirely surrounded by water, no portion being more than 50 miles from the sea . . . has 4625 miles of coastline, guarded by 356 lighthouses and fog alarms. The province is renowned for its picturesque coastline, its harbors dotted with islands, its deep valleys and multitude of beautiful lakes."



Looking across the water from the beach at Ingonish, one can see the Keltic lodge, where the Industrial Minerals Div. will make its headquarters for the September meeting in Nova Scotia.

Cape Smokey looms against the Nova Scotia sky across Ingonish Harbor. In the foreground is the Look-Off at the entrance to the Cape Breton Highlands National Park. Nova Scotia abounds in scenic wonders and facilities for the sportsminded.

A progress report on the plans for the meeting will be made monthly. Thus far, technical sessions will be held at Keltic Lodge, Ingonish, N. S. The lodge will also be the starting point for many of the field trips and social events which are in various stages of planning and development.

Columbia Mines School Offers Student Grants

Columbia University School of Mines announced a scholarship for graduate or undergraduate work, and a graduate research fellowship, in mining engineering, mineral, or metallurgical engineering.

Applications are being taken for the Henry Krumb Scholarships toward undergraduate and professional degrees. Applicants can be either enrolled or in the process of matriculation to the School of Mines, and engaged in undergraduate work toward a bachelor of science, professional degree, or master of science degree. Preference will be given to students in mining engineering.

Applications for the William Campbell Fellowships are open to graduate students in metallurgy, mineral engineering, or mining. About five grants are available, providing stipends of $1800 per year for single, and $2100 for married students.

A complete directory of AIME members, with alphabetical and geographical listing in the form given in the 1948 Directory, but with divisional membership also indicated, will be mailed with the January issues of the three journals. Publication had been scheduled with the December issues, but completion of the Flexoprint installation for publication of the volume was not achieved until mid-November, and a large special issue of MINING ENGINEERING and an unusually early mailing date for the JOURNAL OF METALS in December were contributing factors to the postponement.

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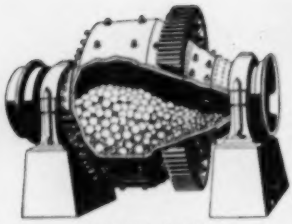
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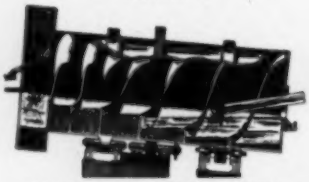
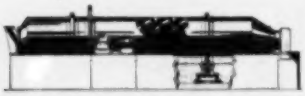
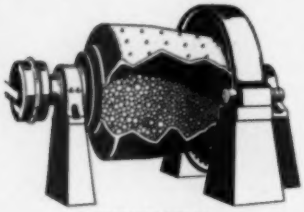
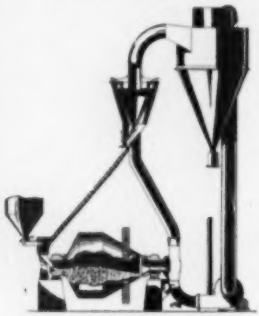

COUNTER-CURRENT CLASSIFIERS
HEAVY-MEDIA SEPARATORS

CONICAL SCRUBBERS

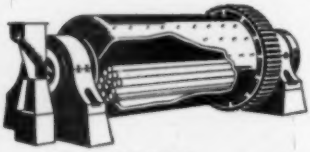

ROTARY COOLERS

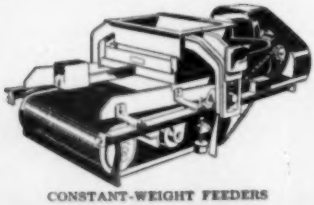
DOUBLE-SHELL ROTARY DRYERS


TRICONE MILLS


AIR CLASSIFYING SYSTEMS

ROTARY AND SHAFT KILNS


ROD MILLS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1953 Publications' Policies Established

Pursuant to Article X of the bylaws of the AIME, the following information is hereby given as to the "conditions, prices, and terms under which the various classes of Members, and Student Associates, severally, shall be privileged to obtain publications of the Institute during the ensuing year."

Publications authorized for issue in 1953 include the following: MINING ENGINEERING, published monthly, containing material, including technical papers, of interest to those engaged in exploration, mining geology and geophysics, and metal, nonmetallic, and coal mining and beneficiation, and fuel technology. The JOURNAL OF METALS, published monthly, containing material, including technical papers, of interest to those engaged in nonferrous smelting and refining, iron and steel, and physical metallurgy. The JOURNAL OF PETROLEUM TECHNOLOGY, published monthly in Dallas, containing material, including technical papers, of interest to those engaged in petroleum and natural gas drilling and production.

Annual subscriptions to any one of the above journals will be provided all members in good standing without further charge. (A member ceases to be in good standing if current dues are not paid by April 1). If more than one of the monthly journals is requested, $4 extra will be charged for an annual subscription, or 75¢ for single copies of regular issues and $1.50 for special issues. The nonmember subscription price for each journal is $8 in the Americas; foreign, $10, and for single issues $1 for regular issues and $2 for special issues. Student Associates will be entitled to the same privileges for all publications as members. AIME members subscribing to more than one of each of the three monthly journals will be billed at the nonmember rate of $8 per year, domestic; $10 foreign, for the extra subscription(s).

Three volumes of *"Transactions"* are authorized for 1953 publication, as follows: No. 193, Mining Branch; No. 194, Metals Branch; and No. 195, Petroleum Branch. These volumes will be available to members at $3.50 each for a first copy if paid for in advance with dues; otherwise at the nonmember rate of $7 less 30 pct. Nonmembers $7 in the United States; foreign $7.50.

Special volumes now planned for publication in 1953 include the following: 1—*Open Hearth Proceedings*, price to AIME members $7; nonmembers, $10. 2—*Blast Furnace, Coke Oven, and Raw Materials Proceedings*, AIME members $7; nonmembers $10. 3—*Electric Furnace Steel Proceedings*, AIME members $7; nonmembers $10. 4—*Dislocations in Metals* (price to be an-

nounced). 5—*Statistics of Oil and Gas Development and Production*, covering data for year 1951, members $5, nonmembers $10. 6—*Statistics of Oil and Gas Development and Production*, covering data for the year 1952, members $5, nonmembers $10. 7—*Symposium on Gasification and Liquefaction of Coal;* price $6 less 30 pct to members. 8—*Problems of Clay and Laterite Genesis;* price $6 less 30 pct to members.

If dues are paid subsequent to January 31, back issues of Institute publications will be supplied only if adequate stocks are on hand. A member may not receive a volume of *"Transactions"*, or a special volume, in lieu of a monthly journal, free of charge on membership. Members in arrears for dues are not entitled to special members' prices for publications.

Rocky Mountain members may have their choice of an annual subscription to one of the monthly journals on request.

1954 Nominating Committee Listed

As provided in Article IX, Section 2, of the AIME bylaws, the names and addresses of the Nominating Committee for Institute officers in 1954 are given below. In each case, the name in parenthesis is that of the alternate, who will serve only in the absence of the principal at the meeting of the Committee.

Mining Branch Council: Newell G. Alford, Alford-Morrow & Assoc., Pittsburgh, Pa. (J. R. Van Pelt, Battelle Memorial Institute, Columbus, Ohio); Grover J. Holt, Cleveland-Cliffs Iron Co., Hibbing, Minn. (J. Murray Riddell, Michigan College of Mining & Technology, Houghton, Mich.).

Metals Branch Council: C. H. Lorig, Battelle Memorial Institute, Columbus, Ohio (G. B. McMeans, Kaiser Steel Corp., Fontana, Calif.).

Petroleum Branch Council: R. W. French, Sohio Petroleum Co., Cleveland, Ohio (John E. Sherborne, Union Oil Co. of Calif., Wilmington, Calif.).

Executive Committee, Council of Section Delegates: W. C. Leonard, Amer. Smelting & Refining Co., Tacoma, Wash. (T. G. Chapman, College of Mines, University of Arizona, Tucson, Ariz.).

Lehigh Valley Section: C. A. Warmkessel, Lehigh Portland Cement Co., Allentown, Pa. (H. R. Gault, Dept. of Geology, Lehigh University, Bethlehem, Pa.).

New York Section: P. Malozemoff, Newmont Mining Corp., New York, N. Y. (F. M. Cameron, St. Joseph Lead Co., New York, N. Y.).

Southeast Section: Garland Peyton, State Div. of Conservation, Atlanta, Ga. (E. H. Rose, Tennessee Coal, Iron & Railroad Co., Fairfield, Ala.).

(Continued on page 1259)

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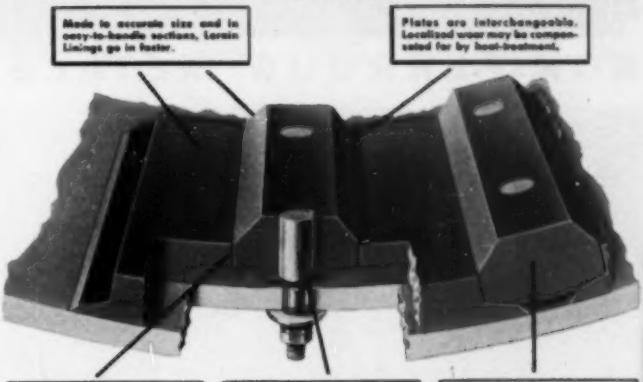
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Plates are interchangeable. Localized wear may be compensated for by heat-treatment.
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Support for bolt head located near bottom of bar . . . bolts are effective for life of lining.
Lift bars designed for correct cascade action. Height of lift varied to suit conditions.

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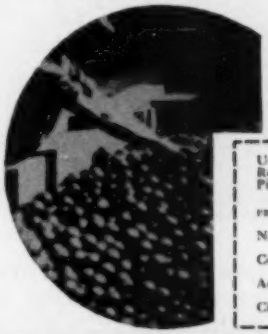
U·S·S Lorain Rolled Plate Linings are available through leading mill manufacturers whose names will be furnished upon request.



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Minnesota Section: D. M. Davidson, E. J. Longyear Co., Minneapolis, Minn. (J. W. Van Evera, Crosby, Minn.).

Utah Section: J. K. Richardson, Salt Lake City, Utah (P. J. Shenon, Salt Lake City, Utah).

San Francisco Section: W. L. Penick, Western Precipitation Corp., San Francisco, Calif. (G. H. Playter, American Smelting & Refining Co., San Francisco, Calif.).

Southern California Section: J. E. Sherborne, Union Oil Co. of Calif., Brea, Calif. (G. D. Dub, Cypress Oil Co., Los Angeles, Calif.).

Gulf Coast: S .E. Buckley, Humble Oil & Refining Co., Houston, Texas (A. W. Waddill, Houston Oil Field Material Co., Houston, Texas).

East Texas: P. J. Lehnhard, East Texas Engineering Assn., Kilgore, Texas (R. K. Thies, Schlumberger Well Surveying Corp., Tyler, Texas).

Members designated by President Haider: S. S. Clarke, chairman of the Committee, Eagle-Picher Co., Cardin, Okla. (F. E. Wormser, St. Joseph Lead Co., New York, N. Y.). Earl R. Parker, Berkeley, Calif. (E. M. Tittman, Salt Lake City). W. H. Geis, Los Angeles (Howard C. Pyle, Los Angeles).

The Committee will welcome suggestions for the official slate for 1954. The following offices are to be filled: President-elect for 1954 (President for 1955), two Vice-Presidents, and six other Directors.

15th Annual Fuels Conference a Success

The 15th annual Fuels Conference, jointly sponsored by the Coal Div., AIME, and the Fuels Div., ASME, attracted 196 registrants to the Bellevue Stratford Hotel, Philadelphia. Cooperating in making the meeting a success were the Philadelphia and Pennsylvania Anthracite Sections of the AIME and the Philadelphia Section of the ASME. A. W. Thorson was the general chairman, while co-chairmen were D. C. Helms of AIME and W. E. Reaser of ASME.

Student members from nearby universities were invited to the banquet, with members of AIME and ASME paying for their tickets. Preceding the banquet, a cocktail party

A. W. Thorson, left, presents H. F. Yancey with the Percy Nichols Award for 1952 at the Fuels Conference Banquet. The award is given for achievement in the field of solid fuels.

was held at the North Garden. A. A. Potter, president of Bituminous Coal Research was the speaker of the evening. H. F. Yancey, Bureau of Mines, Seattle, received the Percy Nicholls Award for 1952.

Highlight of the first day was a luncheon held in the Oak room, with a film, "Powering American Progress", the feature presentation. A Domestic Stoker Symposium program was the first session on the agenda, with several papers presented. Among them were: *Service and Operating Experiences on Two Models of Peroxidizing Domestic Stokers; Comprehensive Studies of Draft Arrangements for Domestic Stokers;* and *New Type of Domestic Anthracite-Burning Equipment.*

Technical papers on the second day were concerned with preparation, and included: *Fundamental Principles Applying to the Design and Operation of a Fine Anthracite Coal Plant at the Coaldale Colliery; Use of Oil in Coal Preparation; Mechanics of Oil Treatment of Coal;* and *A New Method of Spraying Oil at the Georgetown Plant.*

AIME Dues Bills in Mail

Pursuant to Article II, Section 3, of the bylaws of the AIME, notice is hereby given that dues for the year 1952 are payable Jan. 1, 1953, as follows: Members and Associate Members, $20; Junior Members for the first six years of Junior Membership, $12 and thereafter, $17; Student Associates (including an annual subscription to a monthly journal), $4.50.

Mailing of dues bills began in October and will be completed before the end of the year. Prompt payment will assure uninterrupted receipt of the publications desired in 1953. If, for any reason, a bill is not received within a reasonable time after the last mailing date, Dec. 31, 1952, headquarters should be notified.

Legion of Honor — Class of 1903

Anderson, A. J.
Mina, Nev.
Avery, John
Melbourne, Australia

Bruce, James L.
Los Angeles, Calif.

Cautley, A. O.
Northwood, England
Claudet, H. H.
Victoria, B. C., Canada
Clevenger, Galen H.
Centerville, Mass.
Cromwell, Robert H.
East Orange, N. J.

Desloge, Firmin V.
St. Louis, Mo.
du Rell, Charles Terry
Los Angeles, Calif.

Greenidge, S. M.
Fort Worth, Texas

Hamilton, Arthur L.
Lisbon, N. H.

Hooper, Frank C.
North Creek, N. Y.
Huston, Harry L.
Vancouver, Wash.

Jewett, Frank G.
Minneapolis, Minn.

Landers, W. H.
Monterey, Calif.
Leith, Charles K.
Madison, Wis.
Levensaler, Lewis A.
San Francisco, Calif.
Lincoln, Francis C.
Chula Vista, Calif.
Lunt, Horace F.
Denver, Colo.

Marcy, Frank E.
San Diego, Calif.
McClelland, J. F.
New York, N. Y.
Miller, Emory T.
Passaic, N. J.

Moulton, H. G.
New York, N. Y.

Nutter, Edward H.
Saratoga, Calif.

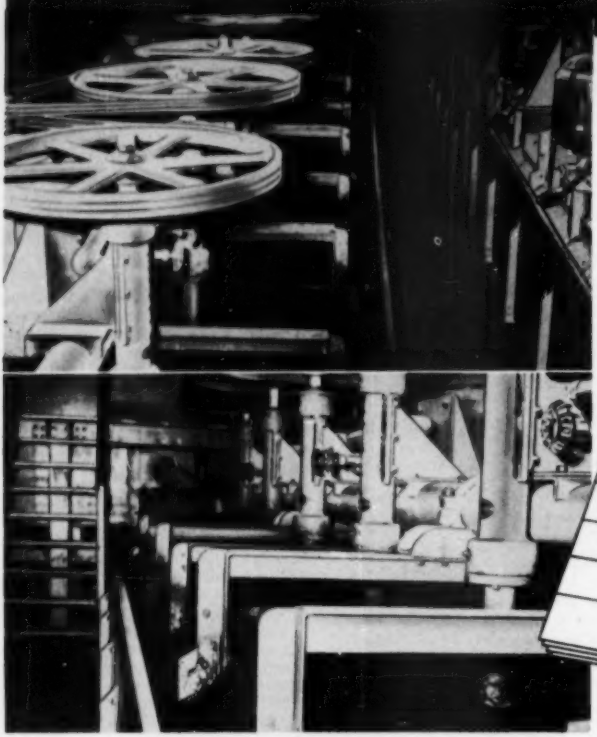
Richardson, Warren Y.
Reno, Nev.

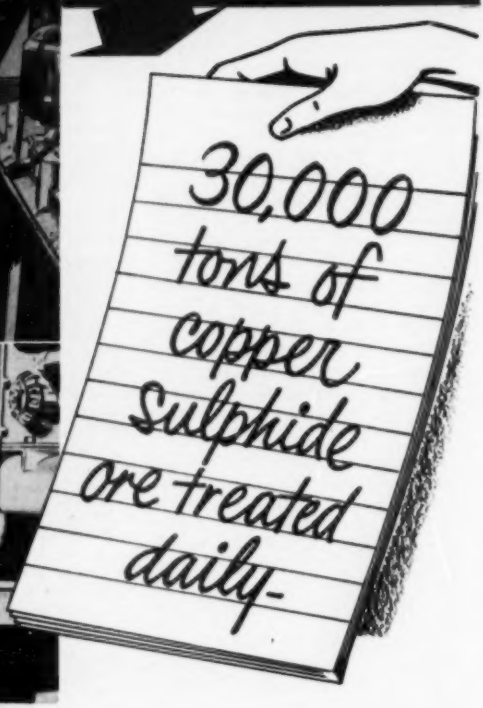
Salsich, LeRoy
Duluth, Minn.
Sherman, Gerald
New York, N. Y.

Throckmorton, Howard
Los Angeles, Calif.

Wainwright, William E.
Melbourne, Australia
Wartenweiler, Fred
Johannesburg, South Africa
Wittenoom, Charles H.
Albany, Western Australia
Wolf, Harry J.
New York, N. Y.
Wraith, William
Long Beach, Calif.

A Statistic like this is never dull...





Why AGITAIR got the assignment

This vast Chile Exploration Co. installation of Agitair Flotation Machines—580 No. 48 cells—is at Chuquicamata, Chile, location of the world's largest copper ore deposit. Agitair was chosen after tests in which the famed Machine led the field in economical use of power and reagent and compared favorably on all other counts. Agitair offers low power consumption, versatility of control, advanced metallurgy, low cost of operation and maintenance and of reagent consumption.

Agitair operates with a minimum of vibration. Froth overflow is easily controlled. There is less wear and tear on mechanism and rubber parts. Easy starting after shutdown and power failure is another feature millmen like.

Write for Bulletin 48 for full data on Agitair and Galigher ore testing service.

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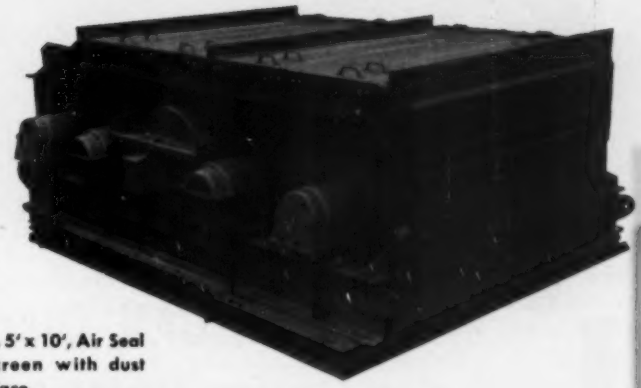
ENGINEERING SERVICE

CONSULTATION • ORE TESTING
PLANT DESIGN • GEOLOGIC INVESTIGATION

18 TY-ROCKS
WILL SCREEN THE ORE AT
CHUQUICAMATA

Ty-Rock Screens will precede the Cone Crushers at the new Sulphide Plant of Chile Exploration Company at Chuquicamata, Chile. The Ty-Rocks will by-pass the under-size ore around the crushers to enable them to operate at peak capacity and efficiency.



Type F-600, 5' x 10', Air Seal Ty-Rock Screen with dust covers in place.

Type F-900, 6' x 14', Air Seal Ty-Rock Screen with covers removed. This rugged screen used in the crushing plant.

Ty-Rocks are preferred by the world's largest mines for their proven high capacity, reliability and efficiency.

THE W. S. TYLER COMPANY

Cleveland 14, Ohio • U. S. A.

WOVEN WIRE SCREENS • SCREENING MACHINERY • TESTING SIEVE EQUIPMENT

AMERICAN Disc-Type FILTERS

dewatering flotation concentrates at

CHUQUICAMATA

Chile Exploration Company

Send for our new metallurgical filter bulletin giving details of the Oliver United line of Metallurgical Filters. Of particular interest will be the two relatively new filters to the list of continuous vacuum filters—Oliver Horizontal Filter and Oliver Panel Filter. Just ask for Bulletin 20-M.

Oliver Continuous Filter . . . the pioneering automatic continuous vacuum filter in the country, brought out in 1907; long considered the standard throughout the mining industry. Standard sizes range from 18 to 790 sq. ft. of filter area.

Dorrco Continuous Filter . . . especially adaptable for dewatering heavy, fast settling solids. The drum acts as the tank with the cake forming on the inside of the drum. Standard sizes range from 18 to 783 sq. ft. of filter area.

CHUQUI

Another Leader Recognizes . . . and Buys Quality in Filter Design, Construction and Service

Here at Chuquicamata, Chile, is the most important new mill in South America, designed to handle daily up to 30,000 tons of copper sulphide ore.

With few exceptions, whenever a 'feature' mill *anywhere in the world* goes into production, you can find Oliver United filters handling the dewatering and/or washing of the milled ores. What is it that so often induces those responsible for the selection of mill equipment, generally in widely separated and distant areas, to rely upon Oliver United equipment? Bearing in mind that there is something of a finality to the purchase of mining and milling equipment to be used thousands of miles and across oceans from the manufacturer, these two values, not to be found elsewhere, are strongly influencing factors:

- **One value is the acknowledged wide experience with and understanding of filtration practices throughout the world. Oliver United's service in this field goes back more than 45 years. Our engineers know ores and their filtration characteristics.**

- **The other value is the wide selectivity in filter types offered by Oliver United. For Chuqui, Americans were selected as being most suitable. Other mills have Olivers or Dorrcos. And now with the Horizontal and Panel Filters entering the filtration picture, each in a wide range of sizes, Oliver United *selectivity* looms even more important.**

Yes, this world-wide preference for Oliver United filtration equipment is basically a matter of *reliance* . . . reliance upon the services and equipment of the company long known in the mining industry as Filtration Headquarters. This reliable service is available to you in connection with your filtration problems.

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Dorr-Oliver S.a.R.L. Milano
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SOUTH AMERICA & ASIA
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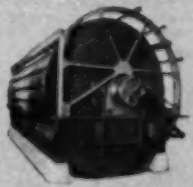
Stamford, Conn.

SOUTH AFRICA
E. L. Bateman Pty., Ltd.
Johannesburg, Transvaal

Oliver Continuous Vacuum Horizontal Filter . . . all essential operations visible to the operator at all times; gravity aids in flow of filtrate; excellent for dewatering and washing crystalline and fibrous products; counter current washing practicable. Standard sizes range from 3.5 to 165 sq. ft. of filter area.



Oliver Panel Filter . . . designed to operate with high capacity when handling pulps producing thin, sticky cakes or pulp with very small percentages of solids. No wire winding, although easily converted to the wire wound drum type of filter. Standard sizes range up to 790 sq. ft. of filter area.

Check up on our Olivite Acid-Handling Pumps and in ODS Diaphragm Slurry Pumps, both quite useful in and around ore-milling plants.

200-ton 6-motor 98'-3" Span Crane with two 100-ton Hoists and one auxiliary 10-ton Hoist. In Concentrator Grinding Bay.

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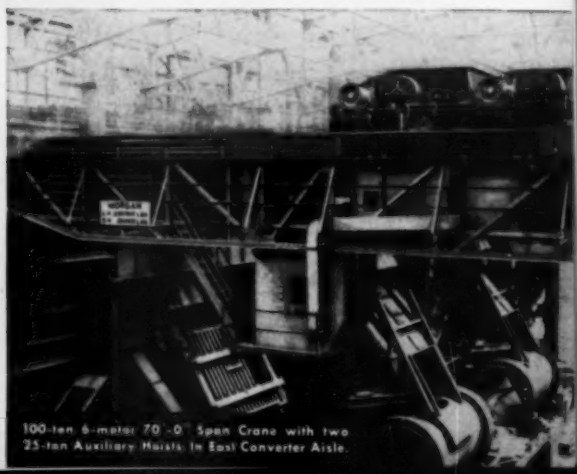
BUILT BY MORGAN Engineering

MORGAN CRANES AT "CHUQUI"

Illustrated are three Morgan Heavy Duty Mill Type Cranes at Chile Exploration Company's new plant at Chuquicamata. These cranes are of very rugged construction, especially designed for copper smelter service. *Your inquiries will be appreciated.*

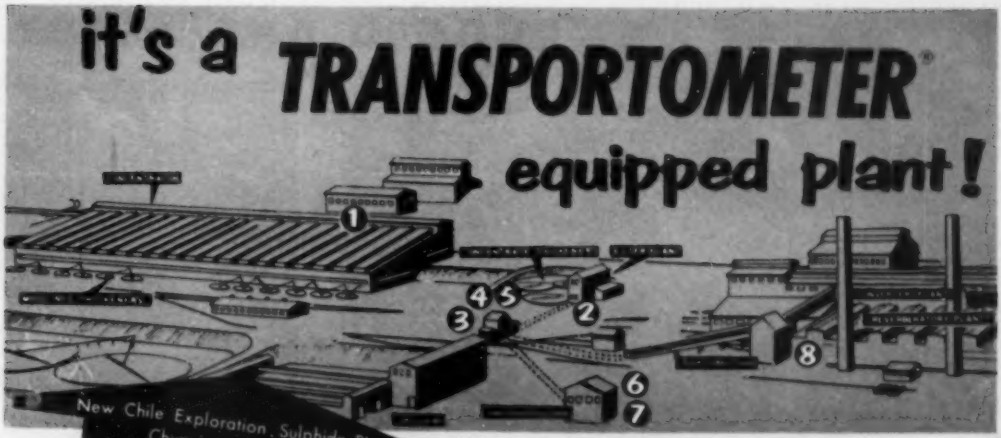
THE MORGAN ENGINEERING CO.
ALLIANCE, OHIO PITTSBURGH—1420 OLIVER BUILDING

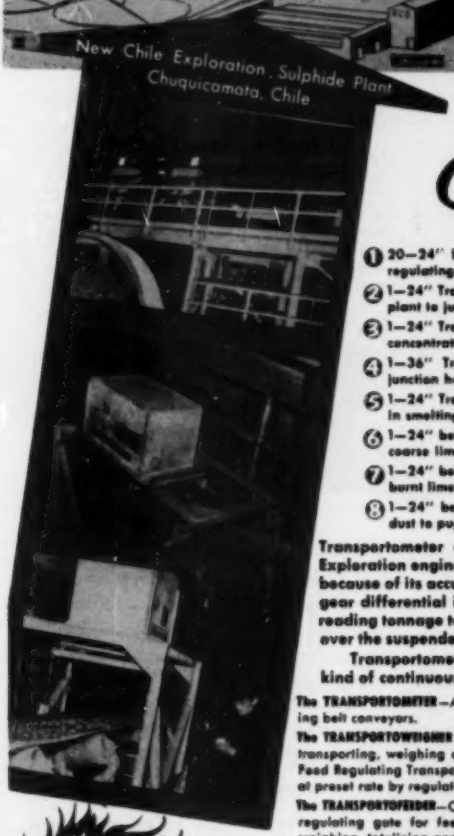
100-ton 6-motor 70'-0" Span Crane with two 25-ton Auxiliary Hoists. In West Converter Aisle.



100-ton 6-motor 70'-0" Span Crane with two 25-ton Auxiliary Hoists. In East Converter Aisle.

it's a TRANSPORTOMETER® equipped plant!





27 Automatic Continuous Weighers

1. 20—24" belt feed regulating Transportoweighers, installed in pairs, for weighing and regulating feed of copper ore to rod mills in concentrator plant. Capacity—160 tons/hour.
2. 1—24" Transportometer for weighing copper concentrate filter cake on conveyor from filter plant to junction house. Capacity—360 tons/hour.
3. 1—24" Transportometer for weighing flux on conveyor from surge bin in junction house to concentrate beds. Capacity—25 tons/hour.
4. 1—36" Transportometer for weighing copper concentrate and flux on conveyor from junction house to smelting plant. Capacity—800 tons/hour.
5. 1—24" Transportometer for weighing siliceous ore on conveyor from junction house to bins in smelting plant. Capacity—500 tons/hour.
6. 1—24" belt Transportofeeder in lime hydrating plant for weighing and regulating feed of coarse lime rock to kiln. Capacity—20 tons/hour.
7. 1—24" belt Transportofeeder in lime hydrating plant for weighing and regulating feed of burnt lime to ball mill. Capacity—30 tons/hour.
8. 1—24" belt Transporteweigher in silica slurry plant for weighing dry reverberatory flue dust to pug mill for reprocessing. Capacity—12 tons/hour.

Transportometer equipment was specified by Anaconda Copper Mining and Chile Exploration engineers for use in the new Copper Sulphide Plant at Chuquicamata, Chile, because of its accuracy and dependability. These units employ a completely mechanical, gear differential integrator which automatically and continuously records on a direct-reading tonnage totalizer, the product of belt speed and material weight per foot passing over the suspended idlers.

Transportometer weighing units are provided in three types for practically every kind of continuous weighing and totalizing required in modern processing plants.

The **TRANSPORTOMETER**—Automatic, continuous weighing and totalizing unit for installation on new or existing belt conveyors.

The **TRANSPORTOWEIGHER**—Complete unit, with its own belt conveyor and drive, for automatic, continuous transporting, weighing and totalizing between conveyor lines, and from conveyor to process equipment. Feed Regulating Transporteweigher is equipped with Electric Switch Mechanism to feed material to weigher at preset rate by regulating preceding belt conveyor, gate, etc.

The **TRANSPORTOFEEDER**—Complete unit, with its own belt conveyor and drive, and a motor operated feed regulating gate for feeding material at preset constant rate. For automatic, continuous transporting, weighing, totalizing and feed regulating from hoppers, bins, etc., to conveyors or process units.

If you have weighing-conveying problems, let us show you what Transportometer equipment can do to solve them.

TRANSPORTOMETER DIVISION

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RUGGED AND READY

anywhere ... anytime

This husky six-wheel Sterling-White really makes the ore fly at the Chuquicamata operations of Chile Exploration Company ... typical of Sterling-White dependability around the world.

No bogging down with Sterling-White. Mountains of dirt really move! Wheel spinning and slipping cut to a minimum because Sterling-White is engineered for more work, for more years ... to its specialized road and load conditions. There is a range of models for all work requirements ... powered for economy and stamina ... for longer ... on the hard jobs.

Here is why STERLING *gets more work done*

- Husky rear axles—single or tandem, in chain drive, enclosed drive and planetary drive models.
- Gasoline or diesel engines.
- Sturdy front axles with ample load capacity for all types of severe service.
- Range of wheelbases for all types and sizes of bodies and other special chassis equipment.
- Unit and auxiliary transmission combinations to meet specialized demands.
- Weatherproof cab, engineered for driver comfort. Flexible 3-point mounting, heavy-duty construction.
- Rear main and auxiliary springs cushion shock and strain, increase tire life, reduce driver fatigue.
- Power steering available in most models.

Sterling White

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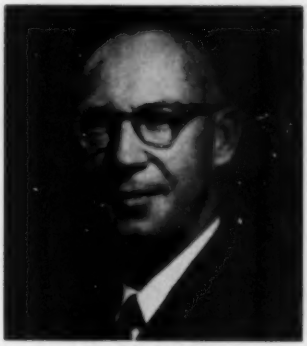
THERE IS A DEPENDABLE WHITE FOR EVERY TRANSPORTATION SERVICE

R. S. NEWLIN



T. A. CAMPBELL

BURR WHEELER

RODOLFO MICHELS

Thomas C. Baker, Jr., has been transferred by the American Manganese Steel Div. of the American Brake Shoe Co., from their Oakland, Calif., branch to their Cleveland, Ohio, office. He will represent them in Ohio and Michigan.

William H. Carson, has been appointed district manager of Goodman Mfg. Co's. Huntington, W. Va. sales office. He succeeds **Morris W. Highland,** granted a leave of absence for health reasons. Mr. Carson, a graduate engineer of Michigan College of Mining & Technology, has been a member of the company's Huntington sales force since 1939, the past two years as assistant district manager.

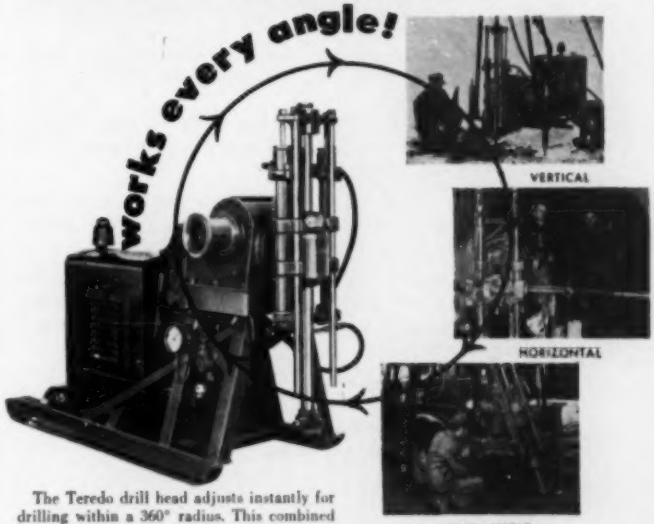
James Parnell Caulfield, former general manager of the Utah Copper Div. of Kennecott Copper Corp., has been appointed assistant general manager of Kennecott's Western Mining Div. **L. F. Pett,** general supervisor of operations of the Utah Copper Div., was named Mr. Caulfield's successor as general manager.

William D. Charles, on completion of his graduate studies at the Massachusetts Institute of Technology, accepted the position of mineral engineer with American Cyanamid Co.

Richard S. Newlin, formerly assistant vice president in charge of mining operations, has been elected vice president in charge of mining operations of Anaconda Copper Mining Co., succeeding **Clyde E. Weed,** who was recently elected vice president in charge of operations of the company and its subsidiaries. **Thomas A. Campbell,** who has been vice president and managing director of the Chilean subsidiaries, has been elected executive vice president of the Chilean subsidiaries to succeed **Burr Wheeler** who is retiring after 38 years of service. **Rodolfo Michels,** vice president of Andes Copper Mining Co., and Chile Exploration Co., has been elected vice president and resident director of those companies with headquarters at Santiago, Chile. **Henry E. Gardiner** of Santiago, has been named assistant vice president of the two Anaconda subsidiaries.

Alan T. Broderick has resigned as regional geologist with the M. A. Hanna Co., and to accept the position of chief geologist with the Inland Steel Co.

The NEW Acker Teredo Core Drill . . .



The Teredo drill head adjusts instantly for drilling within a 360° radius. This combined with other Acker features such as built-in pumps, cargo-type winches, hydraulic or screw feed heads, choice of power plant, mounting on drag or any type of vehicle, are bound to save time, labor and on-the-job costs.

See for yourself—write for Bulletin 33, ME

Over 30 years' experience in the design and development of Drilling Equipment.

ACKER DRILL CO., INC.
SCRANTON 3, PA.

Glenville A. Collins, mining engineer with offices in Santa Barbara, Calif., and at Vancouver, B. C., has recently returned from examinations on Rocher Deboule mountain near Hazelton, British Columbia, and will spend two months in Santa Barbara before making mine inspection trip in Mexico and Central America.

Professor H. Edward Flanders, professor of metallurgy, University of Utah, won second place prize of $300 for a research paper on welding in the International Prize Competition sponsored by Eutectic Welding Alloys Corp. of New York. His paper dealt with the use of low melting alloys in welding.

A. F. Frederickson has returned to Washington University, St. Louis, Mo., after a 14 month stay in Europe. He held a Fulbright Research Fellowship at the University of Oslo for 9 months and visited some of the classic mining regions of Norway, Sweden, Spain and North Africa. As associate professor of geology at Washington University, he will teach geochemistry and petrology.

Chester Freshour is now employed by the Shattuck Denn Mining Corp. He was formerly employed by Phelps Dodge Corp.

Ellis H. Gates has recently joined the staff of the Kennecott Research Center at Salt Lake City, Utah.

J. F. Frost was transferred to the Western Mining Dept. of the American, Smelting, and Refining Co., Salt Lake City.

Clyde W. Hall has retired as president of United Clay Mines Corp., Trenton, N. J., and has become chairman of the board.

Henry K. Martin has been named manager of the Mining and Milling Machinery Div. of the Lake Shore Engineering Co. A graduate of the University of Minnesota, Mr. Martin has been active in the mining and milling industry since 1926.

Ely Mencher has resigned his position as research geologist for Socony-Vacuum Oil Co. of Venezuela to become associate professor at Massachusetts Institute of Technology.

L. R. Messer, consulting mining engineer, will be at Almaden Mine in Spain supervising the erection of modern ore reduction plants designed and furnished by Pacific Foundry Co., Ltd., San Francisco.

Arnold H. Miller, consulting engineer of New York, has been retained by the Phoenix Primera Lead-Zinc Mines, at Lajal, Manzanillo, Oriente, Cuba doing examination and planning work.

James R. Miller has recently accepted a position with the Link-Belt Co. in Chicago, Ill.

Clarence O. Mittendorf, acting head of the Defense Minerals Exploration Administration since its establishment last fall, has been named administrator of that agency, Secretary of the Interior, Oscar L. Chapman announced. Mr. Mittendorf has had a life-long association with the minerals industries.

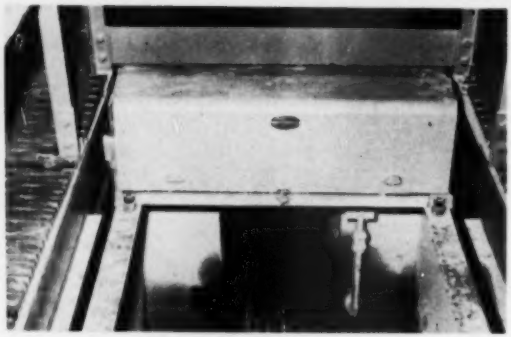
Jose V. Monfort is now with Itogon Mining Co. as mill shift boss at their cyanidation plant, Philippines.

R. E. Nelson, engineer and former assistant to the president at American Gilsonite, was named production manager. Mr. Nelson, a graduate of the Massachusetts Institute of Technology joined the firm in 1948, leaving Standard Oil of California where he had held supervisory posts.

John C. Nigra, who recently resigned as research geologist for the Arabian American Oil Co. in New York City, has accepted an appointment as associate professor of geology, Tulane University, New Orleans, La.

A. L. Payne, formerly with American Smelting & Refining Co.'s Western Mining Dept., is now at Minas de Matahambre in Cuba.

Some Reasons Why it's GEARY-JENNINGS!



One of 52 Geary-Jennings Samplers operating at CHILE EXPLORATION CO., Chuquicamata, Chile.

The Geary-Jennings Sampler has won its way to the top by demonstrated efficiency in metallurgical industries throughout the world.

Comparative tests lead engineers unerringly to the Geary-Jennings true principle of sampling . . . "all of the stream part of the time."

Readily adaptable to small or large operations — wet or dry materials. Adjustable to radical changes in stream composition. Easy interval changes. Controls that prevent human error or tampering.

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Salt Lake City, Utah
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EASTERN OFFICE
921 Bergen Avenue
Jersey City
New Jersey

Agents in All Principal Foreign Mining Districts

G. B. MacKenzie has been appointed mine manager for the Abbeytown Mining Co., Ltd. in Eire.

P. K. Rains has joined the Joy Mfg. Co. and is presently located at the Michigan City plant. Mr. Rains graduated in 1952 from the University of Utah. He is distinguished by receiving the Old Timers Award, given to outstanding students in coal mining.

Richard D. Riddell is employed by the Phelps Dodge Corp. at Bisbee, Ariz. as a transitman.

Richard G. Amidon, superintendent of Buffalo-Boston and Tillieum mines near Granite, Ore., has been in New York City on mining business.

Arthur W. Brune, who recently received a Ph.D. in mining engineering from Pennsylvania State College will be an assistant professor at Lehigh University, working under Dr. R. T. Gallagher.

Stuart St. Clair, consulting geologist and engineer of New York, left the middle of November for a three months professional trip in Asia.

Francis O. Case, vice president of Anaconda Copper Mining Co., was elected president of the company's new subsidiary, The Anaconda Aluminum Co. During World War II, Mr. Case was general manager of Basic Magnesium, Inc. Mr. Case was at one time manager of midwest district of International Smelting & Refining Co. and manager of Anaconda Sales Co.

A. F. Banfield of Behre Dolbear & Co. has been directing a geological survey field party in Ontario during the summer. He expects to return to New York about October 15th. **Park A. Hodges** of this company is making mine examinations in South America. **Dr. Charles A. Behre** has returned to Columbia University after a summer spent in western United States and Mexico.

Martin Fitzimmons will be general manager of Cyanamid de Mexico and in charge of its sales of chemicals for the mining industry. Most recently he was with American Cyanamid Co's. research laboratories at Stamford, Conn., as a mining engineer.

Mine Reports

Requests for reprints of the article "Mine Reports" *(Mining Engineering* August 1952, page 764) from colleges having mineral engineering curricula have been so profuse that it has been decided to make separate copies available to these institutions. Author Samuel H. Dolbear has filled many of these requests from his supply. Additional requests should now be directed to *Mining Engineering,* 29 W. 39 Street, New York 18, N. Y. and orders up to 50 copies per college will be filled free of charge.

Special Bulletins for Mining Engineers!

Get These Facts on NAYLOR Light-Weight Pipe and NAYLOR Wedge-Lock Couplings

Bulletin No. 507 presents the many applications of NAYLOR Pipe in mining service, together with complete specifications, fittings and connections. It is a helpful guide for mining engineers.

Bulletin No. 513 gives the facts on the NAYLOR Heavy-Duty Wedge-Lock Coupling and tells how this one-piece, positive-type connection saves time, work and money on pipe line installation.

Write for these bulletins today.

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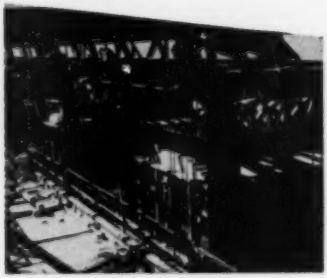
Chuqui

is C-E equipped



Building a plant as huge as Chuquicamata at 9,200 feet above sea level is a tremendous job — among other reasons because it is so difficult to get material to the site.

Chuqui's boiler plant presented problems, too. The eight C-E waste heat boilers had to be designed to handle 40% greater gas volume than at sea level, because of the rarefied atmosphere at 9,200 feet. Since these boilers operate on natural draft, using as a heat source the sulfur-laden waste gases from the copper reverberatories, their stacks soar skyward for 300 feet. This height provides adequate draft for boiler operation . . . and also assures removal of these waste gases from the plant site.

Because of the cooperative efforts of Wilbur Jurden, Chief Engineer of the Anaconda Copper Mining Company . . . and Bruno F. Koch, Anaconda's Power Plant Engineer who worked closely with officials of Chile Exploration and C-E Engineers . . . the design of the Chuqui boilers is indeed tailored to their unusual operating conditions.

The selection of Combustion Boilers for Chuquicamata reflects C-E experience in this highly specialized field . . . an experience backed by service-proved installations in some of the world's foremost copper refineries.

Throughout all industry — large or small . . . domestic or foreign — C-E Equipment is the frequent choice among those who demand efficient, low-cost, reliable performance from the steam generating equipment they buy. B-432

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COMBUSTION ENGINEERING — SUPERHEATER, INC.

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ALL TYPES OF STEAM GENERATING, FUEL BURNING AND RELATED EQUIPMENT

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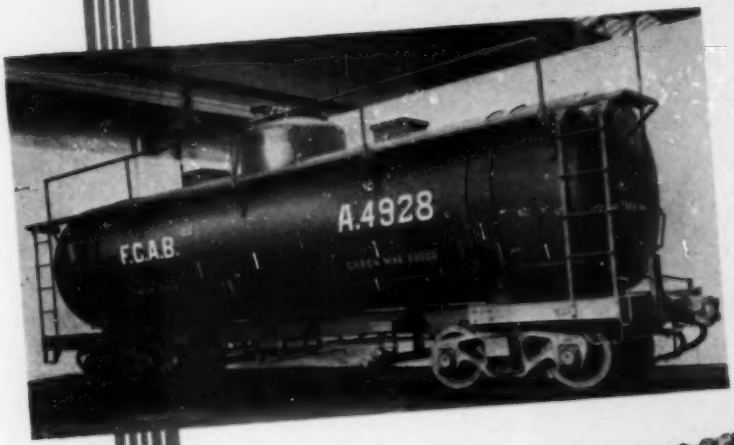
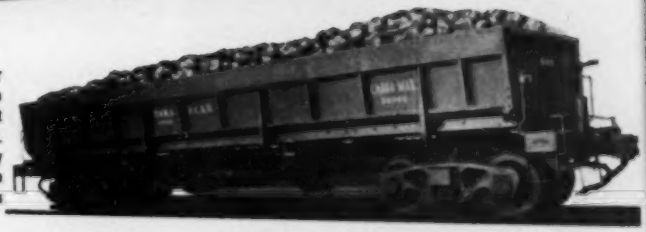


TANK CARS, built to conform to customer's specification, of capacity, track gauge, draft gear and brake, as desired.

DUMP CARS, either automatic air dump type or without dumping cylinders. Built to any capacity or track gauge to suit customer's specification.



Illustrated are two of the many types of freight cars built by The Gregg Company, Ltd., for railroads throughout the world.

Gregg Company engineers, through periodic visits to our many foreign customers, thoroughly familiarize themselves with the particular needs of the various industries of the world, including MINING. Our engineering department will design complete passenger and freight cars to your requirements and specifications as to type, size, capacity, track gauge, etc.

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20 YEAR SUCCESS STORY

Action photographs taken at Chuquicamata

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At this impressive ore deposit in Chuquicamata, Chile, Differential Air Dump Cars are used exclusively.

From an initial order for 20 cars (shipped in 1929) the Chuquicamata fleet of Differentials has grown to 120 cars. (And 60 additional cars are on order.) All dump cars purchased for this mine since the initial order in 1929 have been Differentials.

Today's Differential Air Dump Car shows some refinements but sticks to the original double-trunnion, double-fulcrum design which has earned a reputation for speedy, trouble-free and satisfactory performance through the years.

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STEEL CAR COMPANY

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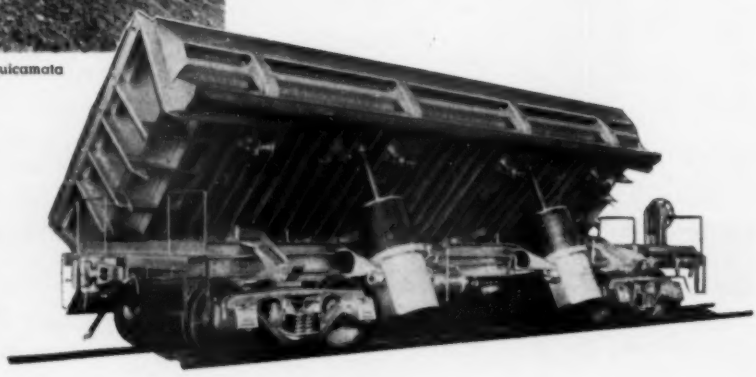

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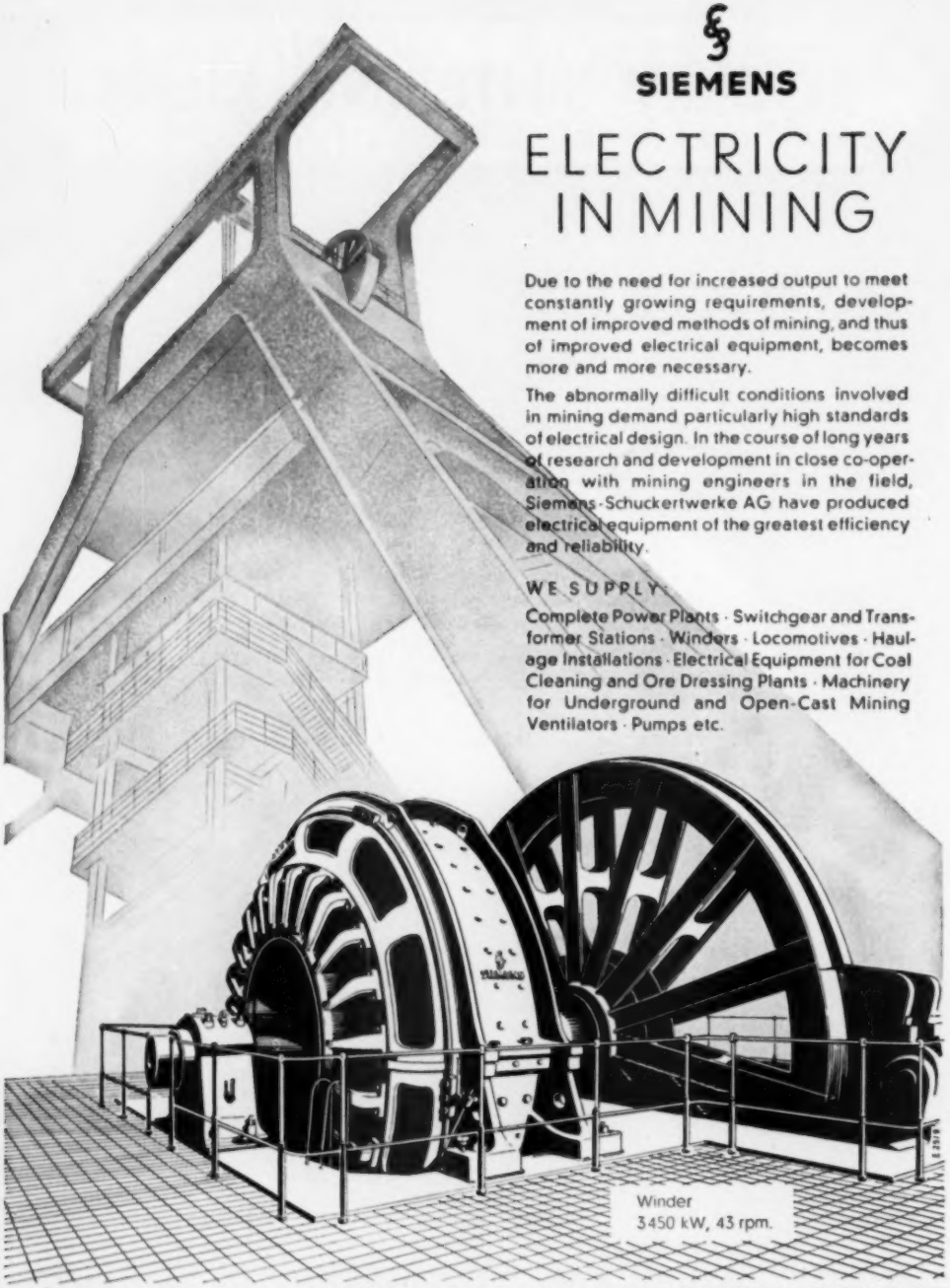

Winder
3450 kW, 43 rpm.

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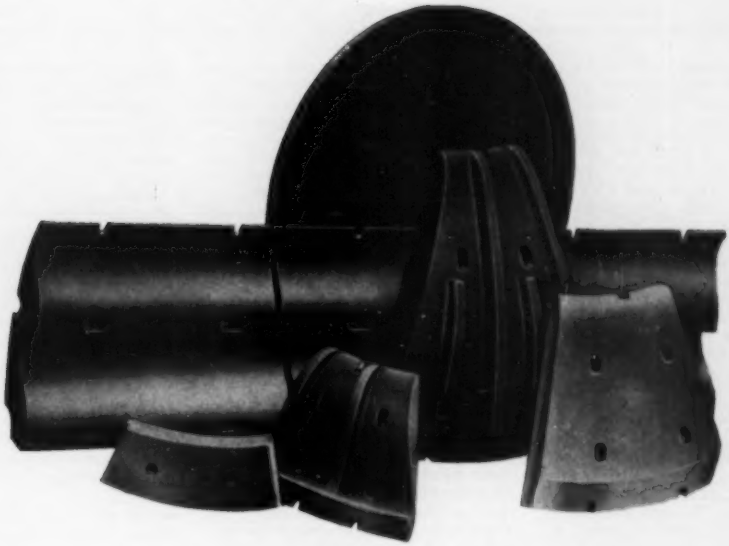

Liners that cross the Oceans

Year after year for use by the mines of the world large quantities of Edgar Allen liners for tube, ball and other mills cross the oceans. The illustration above shows a few IMPERIAL AUSTENITIC MANGANESE STEEL LINERS forming part of a large order for the biggest South American copper mine, the Chile Exploration Co.'s plant at Chuquicamata, Chile. Liners in chromium alloy steel of special hard wearing composition are also supplied, as well as many other types of castings for mines, such as jaws for crushers and granulators, roll shells, sheave wheels, skip wheel, dredger parts, etc., etc.

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Obituaries

Appreciation of
H. M. Hartmann
By B. L. Sackett

In the death of Henry Morris Hartmann, on Aug. 17, 1952, the mining fraternity in the West lost an old friend and an able mining engineer and executive. From 1909, when he was graduated from Sheffield Scientific School at Yale University, to 1945, when he retired as a result of ill health, Mr. Hartmann was prominently identified with a number of well known, western mining enterprises. Starting his professional career with Senator W. A. Clark in Butte, Mont., Mr. Hartmann remained in the employ of the Clark interests for a period of twenty years, during the last thirteen years of which he was general manager for the Ophir Hill Consolidated Mining Co. at Ophir, Utah.

Following five years of private consulting practice, he was associated with the International Smelting & Refining Co. for a period of eight years, (1935–1942), during which time he was managing mining development work at a number of their properties in Nevada and Idaho. From 1938 to 1942 he was, in turn, manager of operations for the Mountain City Copper Co., in Nevada, and for the Walker Mining Co., in Plumas County, Calif. During World War II, he rendered very valuable service to the industry in the West by ably serving as a member of the Nonferrous Metals Commission of the War Labor Board. Mr. Hartmann married Laurie Messias in Butte, Mont., in 1912. He is survived by Mrs. Hartmann and one son, Ernest, of Spreckles, Calif.

Appreciation of
Robert Gale, Jr.
By William A. Kelly,
F. W. Strandburg, R. D. Piper

Robert Gale, Jr., 43, a member of the Institute since 1947 and a mining engineer for the Anaconda Copper Mining Co. since 1939, died Sept. 6, 1952, after a short illness.

Mr. Gale was graduated from the Montana School of Mines in 1934 and spent the first summer thereafter as an instructor at the summer session in surveying. He spent the next four years as a millman, office engineer, and a steel and concrete foreman for the Army Engineers at Fort Peck. He entered the employ of the Anaconda Co. in 1937 and worked as a miner and sampler before joining the engineering staff. Promotions came to Bob Gale because of his ability and unusual capacity for accomplishment. Everyone liked Bob and welcomed him to their offices and mines.

In September 1951, he was given charge of all the engineering work at the Kelley Mine. He was a tireless worker and gave inspiration and encouragement to the men under him.

Bob's associates in the engineering dept. were saddened by his untimely death and will miss his ready smile and friendly generosity. It will be difficult to replace him. He gave promise of bigger things. He was a member of the Montana Society of Engineers, and a prominent member of the School of Mines Alumni.

Appreciation of
Andrew B. Crichton
By Dean Edward Steidle

Andrew B. Crichton died unexpectedly on the morning of July 9 at his home in Johnstown, Penna. following a full day spent at his office working on the many varied affairs with which he had been concerned. A mining engineer and coal operator, Mr. Crichton had interests in other business fields and was extremely active during his entire life in civic and charitable affairs locally and nationally.

President of the Johnstown Coal & Coke Co. which he founded forty years ago, Mr. Crichton was also president of The Crichton Co., a family investment and mining concern; Penvir Realty Co. and Super Service Motor Freight Co., a Tennessee firm in the motor carrier industry.

Born March 4, 1882 in Arnot, Tioga County, Pa., of Scotch ancestry, Mr. Crichton was, in every sense, a self-made man. He entered coal mining as a trapper boy at the age of eleven, following the footsteps of his father, William, who served as mine superintendent for various companies in Pennsylvania and West Virginia during his lifetime. Young Andy worked his way upward, depending upon self-education, correspondence school and an avid desire for knowledge.

He opened his own mining engineering office in Beaverdale, Penna. in 1900. The same year he took up residence in Johnstown. He opened an office in Johnstown in 1907 and first engaged in coal operations a few years later.

His membership in AIME dates from 1914, but his most important achievement as a member of the association resulted from a paper delivered to the New York meeting in 1948 in which he challenged U. S. Geological Survey estimates of the coal reserves of the United States. This paper and other of his writings and efforts are largely responsible for initiation of the current re-evaluation of coal reserves being undertaken by the Government with the assistance of other interested groups.

Having studied the matter of mineral reserves for many years, Mr.

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TABLE

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Crichton qualified himself as an expert on this and other matters partly as a result of his long and varied experience in mining problems throughout the world. In 1921, he was sent to Great Britain and other European countries by the Pennsylvania Railroad. He was charged with the job of investigating mine drainage and stream pollution problems.

Mr. Crichton went to Peru, South America, to inspect mines for a group of American investors in 1923. In 1935, he traveled to Russia to investigate the underground gasification of coal in the Don Basin coal fields.

In 1949, he was named by Secretary of Interior, Julius A. Krug, as one of a panel of 19 to represent the United States in the world scientific conference held by the United Nations at Lake Success. Former President Herbert Hoover, an old friend of Mr. Crichton's, also was on the committee.

Mr. Crichton was chairman of the coal resources committee of the National Bituminous Coal Advisory Council. For many years he was active as a director of both the National Coal Assn. and the Central Pennsylvania Coal Producers Assn.

Energetic in everything he attempted, Mr. Crichton was long a respected leader in his community in church, civic and business affairs. Prominent as a founder of the Westmont Presbyterian Church in Johnstown, he served as an elder for many years. Red Cross, Community Chest, hospital boards and countless other civic activities benefited from his active participation and generous contributions. A director and vice president of the United States National Bank in Johnstown, he was active also in Chamber of Commerce work and served as a director of the Pennsylvania Chamber of Commerce.

In addition to this great variety of outside interests, Andrew Crichton was most intensely a family man. One of ten children, his role was ever that of family protector. As busy as he might have been, he always had time for assistance and counsel to brothers and sisters, any of his own family of six children, countless nieces and nephews or any person in need.

An unusual and wonderful man has left us.

J. L. Rutledge (Member 1905), recently retired director of the Maryland Bureau of Mines, died Sept. 13, 1952 in Baltimore. Dr. Rutledge, who was 85, was a native of Illinois and had spent his entire career in and with the coal industry. He was associated with individual producing companies and the United States Bureau of Mines for many years and with the Maryland Bureau until his retirement several months ago. Dr. Rutledge was born in Illinois and attended the University of Illinois where he received a B.S. in mining.

McHenry Mosier (Member 1936), a mining engineer with the Bureau of Mines, Dept. of the Interior, since September 1935, died on October 19 at Suburban Hospital, Bethesda, Md. At the time of his death, he was executive officer of the Region VIII Bureau of Mines-Geological Survey Field Team passing upon mineral exploration projects for the Defense Minerals Exploration Administration. Except for short assignments at Duluth, Minn., and Pittsburgh, Pa., when he first joined the Bureau, Mr. Mosier was at College Park, Md., and Washington, D. C., throughout his service. Working out of College Park, he supervised a group doing scientific research to advance mining practice. For a time during the war, he was successively assistant chief of the former Mining Division and on the staff of the Assistant Director as chief of the Copper Branch at Washington. When the Bureau was regionalized after the war, he was made chief of the Mining Development Branch of the Region VIII Minerals Technology Division. He is survived by a wife, a son, and two daughters.

NECROLOGY

Date Elected	Name	Date of Death
1903	W. T. Burns	June 25, 1948
1940	Hugh J. Fraser	Aug. 22, 1950
1929	John Hudson Hall	Aug. 14, 1952
1951	W. L. Odom	Aug. 3, 1952
1918	Donald A. Powell	Oct. 13, 1952
1903	Henry S. Young	Unknown

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This man should have a wide acquaintance in the non-ferrous field, capable of contacting customers for engineering and construction contracts involving crushing, grinding and concentrating plants in the ferrous and non-ferrous field. Should also be capable of consulting and working with our engineering department when designing.

Also must be capable of starting up plants which we build.

Salary and participating in profit-sharing bonus.

Write and furnish full details of experience, background, education, and salary required.

Box H-19 MINING ENGINEERING

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Compression Grip distributes strain over whole plate area

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★ Trough naturally, operate smoothly through take-up pulleys.
★ Distribute strain uniformly.
★ Made of Steel, "Monel," "Everdur." Also "Promal" top plates.
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Proposed for Membership

MINING BRANCH, AIME

Total AIME membership on Oct. 31, 1952 was 16,333; in addition 1,897 Student Associates were enrolled.

ADMISSIONS COMMITTEE

T. D. Jones, Chairman; Thomas G. Moore, Vice-Chairman; H. S. Bell, F. W. Hanson, R. H. Chadwick, T. W. Nelson, C. A. R. Lambly, John T. Sherman, A. C. Brinker, G. P. Lutjen, Ivan Given, E. A. Prentis, C. Leslie Rice, Jr., and J. H. Scaff.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

Alabama
Birmingham—George, Paul E. (M)
Birmingham—Hodges, Clarence M. (M)
Birmingham—Sullivan, Samuel E. (M)
Mountain Brook—Kerley, Henry E. (M)

Arizona
Globe—Spencer, John J. (M)
Phoenix—Campbell, Gilbert E. (R. M)

California
San Francisco—Kundert, Charles J. (J)

Colorado
Denver—Campbell, Russell H. (J)
Denver—Thomas, Lester C. (M)
Gilman—Otto, Richard H. (J)
Gilman—Williams, William B. (C/S—S-J)
Pueblo—Weddle, Jesse L. (C/S—S-A)

Idaho
Carey—Albrethsen, Adrian E. (C/S—S-J)
Osburn—Ashwill, Walter R. (J)

Illinois
Park Forest—Walters, William M. (A)

Kentucky
Wheelwright—Shockley, Richard R. (J)

Michigan
Ironwood—Hjalmquist, Walter C. (C/S—S-J)
Ishpeming—Sundeen, Stanley W. (R. M)
Monroe—Meyers, James P. (C/S—S-J)

Minnesota
Aurora—Johnson, Earl R. (R. C/S—J-M)
Hibbing—Bailey, Carlton D., Jr. (C/S—S-J)
Virginia—Rathbun, Grove A., Jr. (C/S—S-J)

Montana
Butte—Lacy, Edward E. (A)
Butte—Pickett, Norman S. (C/S—S-J)

Nevada
Reno—Morrice, Edward (R. C/S—S-J)

New Jersey
Succasunna—Auvil, Jesse H., Jr. (R. C/S—S-J)

New Mexico
Carlsbad—Casabonne, Paul J. (J)
Carlsbad—Evans, Charles E. (M)
Carlsbad—Tyman, Paul M. (R. C/S—S-M)
Hanover—Conley, Leslie J., Jr. (R. C/S—S-J)
Santa Fe—Carmichael, Virgil W. (A)

New York
Great Neck—Rosenberg, Richard A. (C/S—S-J)
Watervliet—Kreitman, Abe (J)

Pennsylvania
Nanticoke—Rusinko, Frank, Jr. (J)
Palmerton—Kirby, Donald E. (J)
Pittsburgh—Nyquist, Hugo C. (R. M)
Pittsburgh—Ramsden, George (M)
Scranton—Bell, Robert W. (M)
Scranton—Hogan, Howard H., Jr. (R. C/S—S-J)

South Carolina
York—Wilson, Lewis G. (M)

South Dakota
Lead—Rizzi, Theophilus M. (C/S—S-M)

Tennessee
Ducktown—Kopp, Guerdon G. (R. C/S—S-M)
Lenoir City—Sense, John J. (C/S—S-J)

Texas
El Paso—Cole, Cornelius C. (M)
El Paso—Courson, William E. (A)
El Paso—Misner, Howard I. (M)
Presidio—Hinton, George B. (R. C/S—J-M)

Utah
Midvale—Nebeker, James S. (R. C/S—S-M)
Salt Lake City—Cone, John H. (M)
Salt Lake City—Conner, Alan G. (R. C/S—S-A)
Salt Lake City—Hammond, Ernest DeF. (M)
Salt Lake City—Squibb, H. David (A)
Salt Lake City—Uhle, Theo. T. (J)
Tooele—Beck, Russell R. (R. C/S—S-A)

Virginia
Arlington—Newcomb, Edward L. (R. C/S—A-M)

Washington
Seattle—Cole, Robert J. (R. M)

West Virginia
Fairmont—Belton, Arthur E. (M)
Fairmont—Judy, George L. (M)

Alaska
College—Schmidt, George R. (C/S—S-A)

Argentina
Caipe, Salta—Perichon, Luis A. (J)

Australia
King Island—Salamy, Stanley G. (R. C/S—S-J)
Radium Hill, So. Australia—Gilberthorpe, Norman A. (J)

Canada
Copper Mountain, B.C.—Ross, John A. C. (M)
Matheson, Ontario—Baker, Robert D. (M)
Virginiatown, Ontario—Beggs, Robert J. (M)
Toronto—Ogden, W. Michael M. (J)

Colombia
Otu, via Medellin—Emdin, Henry J. (M)

E. Africa
Tanganyika, P. O. Mpanda—Blackburn, Norman M. B. (J)
Tanganyika, P. O. Mpanda—Turner, James F. (J)

Mexico
Mexico I D. F.—McGrath, John M. (M)
Santa Rosalia, B. C/o—Garcia-Quintanilla, Francisco L. (M)
Tarco—Brittain, Dale C. (C/S—S-A)

Peru
Arequipa—Rankin, John D. (M)
Casapalca—Dancuart, Rene A. (J)
Tamboraque-Bevasa—Rios, Alberto (M)

S. W. Africa
Tsumeb—Shiress, Keith S. (M)

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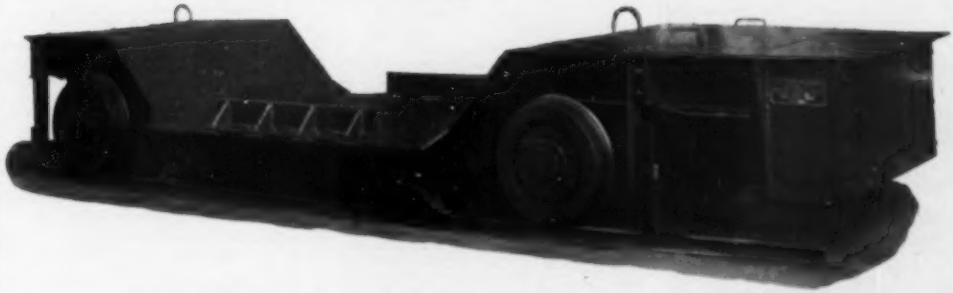
handle Slag Ladles at Chuquicamata



64-ton capacity Slag Ladle Electric Cars. Operated by means of power fed through the track rails which are insulated. The wheels on one rail are insulated and current is picked up by means of collector shoes. The controller and resistors located at the remote operating station.

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—Coming Events—

Dec. 2, American Mining Congress, annual membership meeting, University Club, New York.

Dec. 3, Society for Applied Spectroscopy, Socony-Vacuum Training Center, New York City.

Dec. 4-6, AIME, Electric Steel Furnace Conference, Hotel William Penn, Pittsburgh.

Dec. 5, Atlanta Meeting of Southeast Section, AIME, Ansley Hotel, Atlanta.

Dec. 7-10, American Institute of Chemical Engineers, annual meeting, Hotels Cleveland and Carter, Cleveland.

Dec. 8, AIME, Arizona Section, all-day meeting, Tucson.

Jan. 6, 1953, Society for Applied Spectroscopy, Socony-Vacuum Training Center, New York City.

Jan. 14, 1953, AIME Connecticut Section, Hammond Metallurgical Laboratory, Yale University, New Haven, Conn.

Jan. 14-16, 10th Annual Meeting of Society of Photogrammetry, Shoreham Hotel, Washington, D. C.

Jan. 21-23, Ninth Annual Technical Conference of Society of Plastics Engineers, Inc., Hotel Statler, Boston.

Jan. 26-30, American Society of Heating and Ventilating Engineers, international heating and ventilation exposition, International Amphitheatre, Chicago.

Feb. 4, AIME, Chicago Section, Chicago Bar Assn., Chicago.

Feb. 16-19, AIME, annual meeting, Statler Hotel, Los Angeles.

Mar. 8-11, American Institute of Chemical Engineers, Buena Vista Hotel, Biloxi, Miss.

Mar. 11, AIME, Connecticut Section, Brass Mill Clinic, Waterbury, Conn.

Mar. 16-20, National Assn. of Corrosion Engineers, annual conference and exhibition, Hotel Sherman, Chicago.

Mar. 23-27, ASM western metal congress and exposition, Pan-Pacific Auditorium, Los Angeles.

Apr. 12-May 23, Empire Mining and Metallurgical Congress, Australia-New Zealand.

Apr. 20-23, AIME, National Open Hearth and Blast Furnace, Coke Oven and Raw Materials Conference, Hotel Statler, Buffalo.

Apr. 25-May 10, Fifth Liege International Fair, Liege, Belgium.

May 11-14, American Mining Congress Coal Convention & Exposition, Public Auditorium, Cleveland.

May 14-16, Pacific-Northwest Metals and Minerals Conference of 1953, joint meeting of Metals Branch and Industrial Minerals Div., Ben Franklin Hotel, Seattle.

May 18-22, Fifth National Materials Handling Exposition, Convention Hall, Philadelphia.

June 3-7, Chemical Institute of Canada, Windsor, Ont.

Sept. 15-20, Joint Meeting Industrial Minerals Division, AIME, Keltic Lodge, Ingonish, Nova Scotia.

Sept. 21-23, American Mining Congress Mineral Mining Convention, Olympic Hotel, Seattle.

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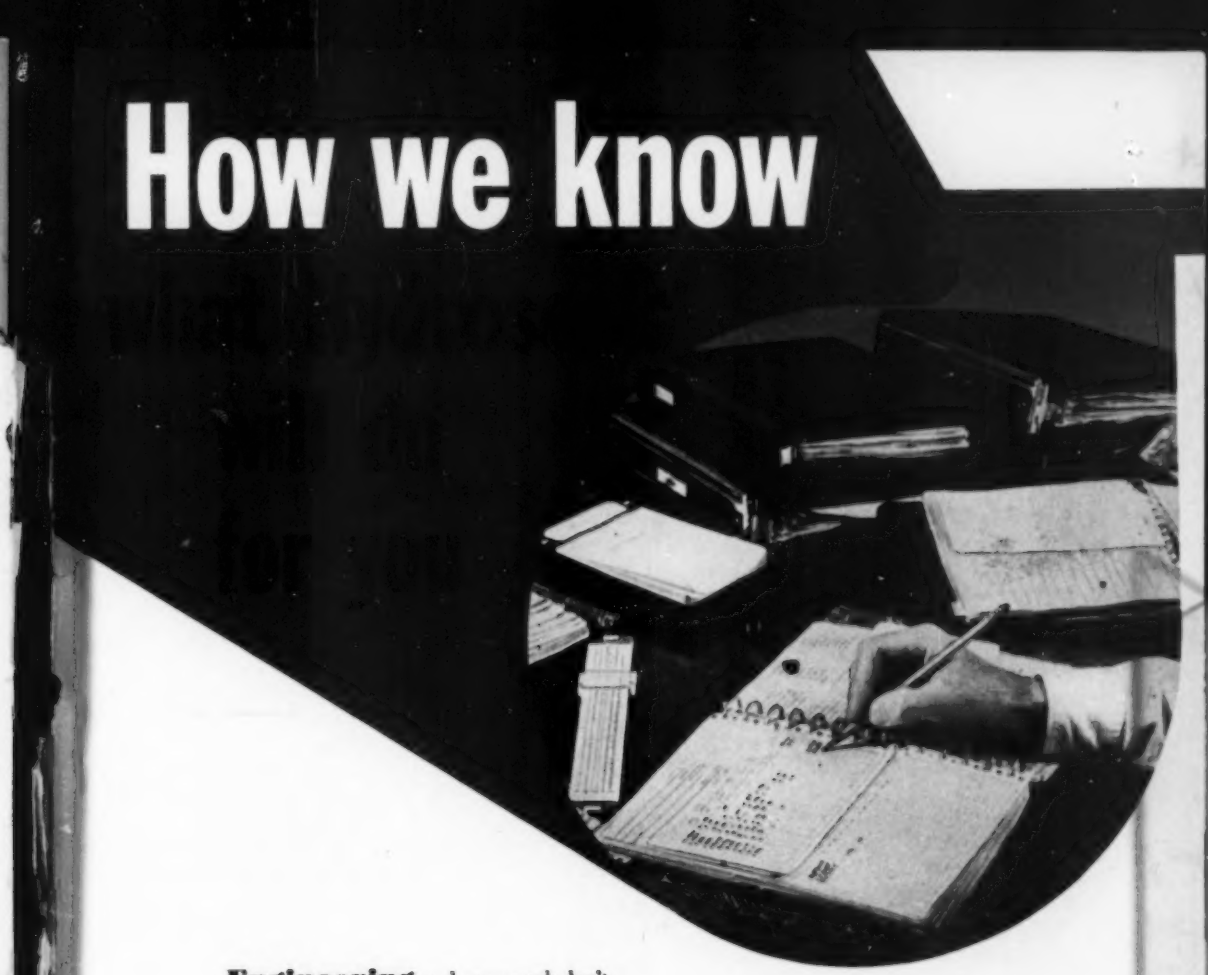

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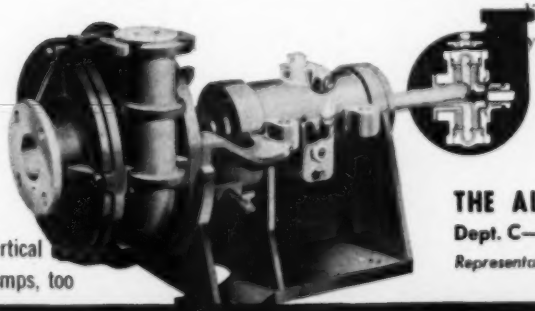

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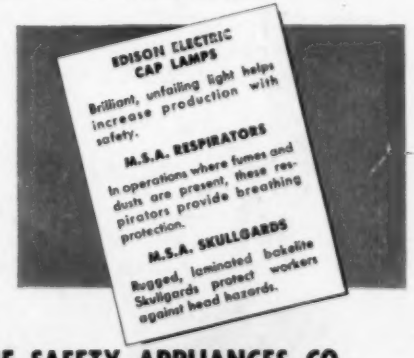
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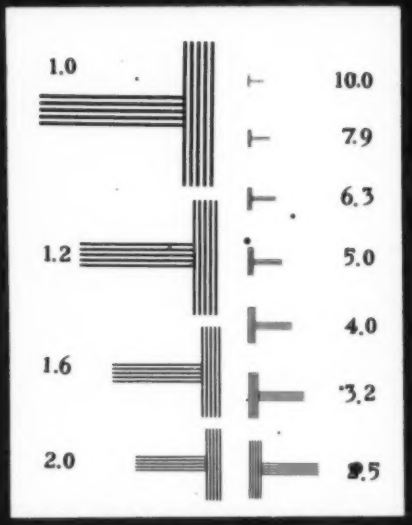

100 MILLIMETERS

INSTRUCTIONS Resolution is expressed in terms of the lines per millimeter recorded by a particular film under specified conditions. Numerals in chart indicate the number of lines per millimeter in adjacent "T-shaped" groupings.

In microfilming, it is necessary to determine the reduction ratio and multiply the number of lines in the chart by this value to find the number of lines recorded by the film. As an aid in determining the reduction ratio, the line above is 100 millimeters in length. Measuring this line in the film image and dividing the length into 100 gives the reduction ratio. Example: the line is 20 mm. long in the film image, and $100/20 = 5$.

Examine "T-shaped" line groupings in the film with microscope, and note the number adjacent to finest lines recorded sharply and distinctly. Multiply this number by the reduction factor to obtain resolving power in lines per millimeter. Example: 7.9 group of lines is clearly recorded while lines in the 10.0 group are not distinctly separated. Reduction ratio is 5, and $7.9 \times 5 = 39.5$ lines per millimeter recorded satisfactorily. $10.0 \times 5 = 50$ lines per millimeter which are not recorded satisfactorily. Under the particular conditions, maximum resolution is between 39.5 and 50 lines per millimeter.

Resolution, as measured on the film, is a test of the entire photographic system, including lens, exposure, processing, and other factors. These rarely utilize maximum resolution of the film. Vibrations during exposure, lack of critical focus, and exposures yielding very dense negatives are to be avoided.

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